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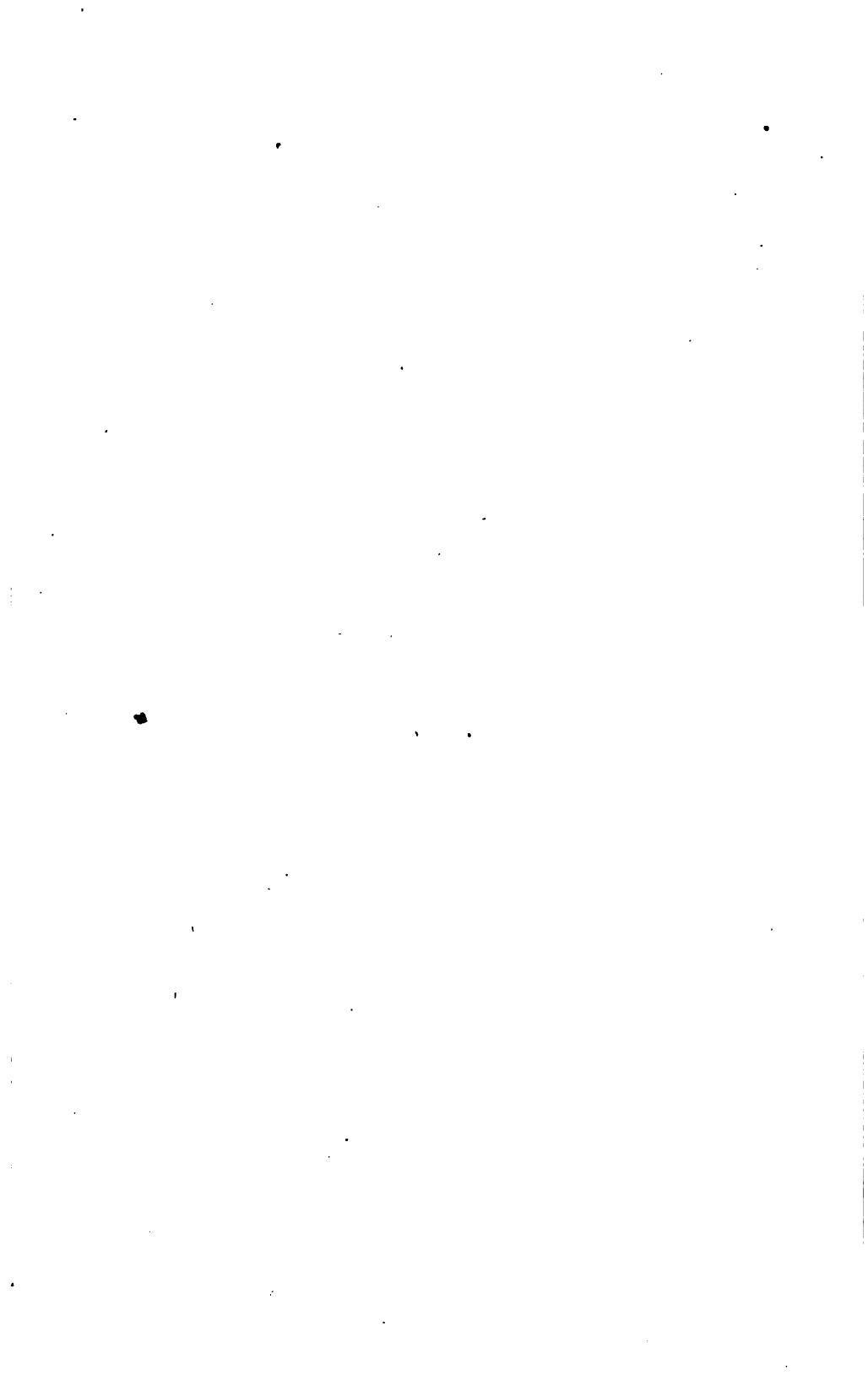
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SEPTEMBER, 1915



Bulletin of the American Institute of Mining Engineers

PUBLISHED MONTHLY

SECTION I

	PAGE		PAGE
Advisory Board for United States Navy..	iii	Library.....	ix
Library Service Bureau.....	iv	Membership.....	xi
Nominations for Officers.....	v	Institute Committees.....	xxiii
Personal.....	v	Officers and Directors.....	xxx
Employment.....	vii		

SECTION II.—PAPERS AND DISCUSSIONS

Important Topping Plants of California. By Arthur F. L. Bell	1789
The Geology of the Iron-Ore Deposits In and Near Daiquiri, Cuba. By James F. Kemp.	1801
Tramming and Hoisting at Copper Queen Mine. By Gerald F. G. Sherman	1837
The Pacific Coast Iron Situation. By Charles Colcock Jones	1887
The Furbero Oil Field, Mexico. By E. DeGolyer	1899
Manufacture and Tests of Silica Brick for the Byproduct Coke Oven. By Kenneth Seaver	1913
Mine Pumping. By Charles Legrand	1929
Conveyor-Belt Calculating Chart. By J. D. Mooney and D. L. Darnell.	1937
Ventilation of the Copper Queen Mines. By Charles A. Mitke.	1941
The Formation of the Oxidized Ores of Zinc from the Sulphide. By Y. Tsenshan Wang.	1959
The Stresses in the Mine Roof. By R. Dawson Hall.	2013
Kick vs. Rittinger: An Experimental Investigation in Rock Crushing, Performed at Purdue University. By Arthur O. Gates	2023
Standardizing Rock Crushing Tests. By Myron K. Rodgers.	2053
Discussion of the Paper of Roswell H. Johnson, The Role and Fate of the Connate Water in Oil and Gas Sands. By C. W. Washburne	2057
Rotary Kilns for Desulphurization and Agglomeration. By Samuel E. Doak.	2061
The Mexican Oil Fields. By L. G. Huntley.	2067
The Cost of Maintaining Production in California Oil Fields. By M. E. Lombardi	2107

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AMERICAN INSTITUTE OF MINING ENGINEERS

29 West 39th Street, New York, N. Y.

PROPOSAL FOR MEMBERSHIP

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(Name in Full)

Occupation _____

Address _____

*is hereby proposed by the undersigned, as a*_____

of the American Institute of Mining Engineers.

Signatures of three
Members or
Associates.

*Place of birth*_____ *Year of birth*_____

*Education, general and technical, when, where and how acquired,
with degrees, if any.*

Dates

[illegible]

[illegible]

Dated _____ 191

ARTICLE II.—MEMBERS.

Sec. 2. The following classes of persons shall be eligible for membership in the Institute, namely: as Members, all professional mining engineers, geologists, metallurgists, or chemists, and all persons actively engaged in mining and metallurgical engineering, geology, or chemistry; as Associates, all persons desirous of being connected with the Institute who in the opinion of the Board of Directors are suitable.

Every candidate for election as a Member, Associate, or Junior Member must be proposed for election by at least three Members or Associates, must be approved by the Committee on Membership, as prescribed in the By-Laws, and must be elected by the Board of Directors.

BULLETIN OF THE AMERICAN INSTITUTE OF MINING ENGINEERS

PUBLISHED MONTHLY

No. 105

SEPTEMBER

1915

Published Monthly by the American Institute of Mining Engineers at 212-218 York St., York, Pa., H. A. WISOTZKY, Publication Manager. Editorial Office, 29 West 39th St., New York, N. Y., BRADLEY STOUTEN, Editor, F. O. PIERCE, Asst. Editor. Cable address, "Aime," Western Union Telegraph Code. Subscription (including postage), \$10 per annum; to members of the Institute, public libraries, educational institutions and technical societies, \$5 per annum. Single copies (including postage), \$1 each; to members of the Institute, public libraries, etc., 50 cents each.

Entered as Second Class matter January 28, 1914, at the Post Office at
York, Pennsylvania, under the Act of March 3, 1879.

ADVISORY BOARD FOR UNITED STATES NAVY

The members of the Institute have probably seen in the daily papers notices of the plans of the Secretary of the Navy to form an Advisory Board to assist the Government to make available the latest inventive genius of the country. Secretary Daniels has appointed Mr. Thomas A. Edison Chairman of this Advisory Board and has officially invited the Institute to secure the selection of two of its members to serve on this Board. The Secretary in his letter states:

"I feel that the work your Institute has done has been such as to give it the right to be in a way officially represented, and the Navy Department desires in this way to testify to its own appreciation of the splendid work for our country that your Institute has done. I have the emphatic approval of Mr. Edison, and he agrees entirely with me that your Institute should be represented in this way and that no better method of getting the kind of men we need could be devised."

The invitation of the Secretary of the Navy has been officially accepted and steps have been taken to nominate the two members. Since the Constitution provides for no meeting of the Board of Directors until September, and as Secretary Daniels urged early action, the matter was discussed at a special meeting of the Executive Committee of the Institute, at which were present as many members of the Board of Directors as were in New York City at the time. Invitations were also extended to other members of the Institute who happened to be in the vicinity. At this meeting it was decided to submit the matter by a special letter-ballot to the Board of Directors and in doing so to suggest or nominate five members of the Institute, accompanying this nomination with the statement that the Directors might vote for two of the members so nominated or for any other members of the Institute not so nominated. It was voted at this meeting that it was the sense of those present that in view of the fact that this committee was an advisory one the type of men to be chosen should be men of administrative or executive ability, cap-

able of passing upon the value of inventions and of obtaining facilities for securing technical advice on questions which might be presented, rather than that the men so chosen should be of distinctive prominence as inventors. No nominations were made at the meeting, but every one was advised to communicate by letter-ballot with the Secretary of the Institute, it being understood that the five names having the highest number of votes should then be transmitted as suggestions to the Board of Directors for their letter-ballot, members of the Board being free to select their own candidates. Nearly all of the 24 members of the Board of Directors have voted, and the Secretary, acting under instructions, has maintained absolute privacy as to the expression of each Director. A meeting of the Executive Committee will be called at an early date, when the two members of the Institute receiving the highest number of votes will be announced and transmitted to the Secretary of the Navy for appointment. The officers of the Institute do not feel free to make any announcement of the names of the members selected until they have received the consent of Secretary Daniels to do so.

LIBRARY SERVICE BUREAU

The Library Board of the United Engineering Society is desirous of bringing closer to you the facilities and usefulness of your splendid Library. To that end, they have inaugurated a Service Bureau whose duty it shall be to maintain such a staff of expert searchers and translators as to insure prompt and thoroughly efficient service to those of our members whose absence from New York makes it inconvenient for them to personally consult the books and periodicals on our shelves.

The Service Bureau is also arranging to keep such members as may desire it, posted on the current publications of any engineering subject in which they may be specially interested.

The Charter of the United Engineering Society does not permit the Library to carry on any work of a commercial character and, therefore, the prices charged members for this service have to be kept down to a merely self-supporting basis.

All the work done by the Library Service Bureau is strictly confidential.

The combined libraries of the American Institute of Mining Engineers, the American Society of Mechanical Engineers, the American Institute of Electrical Engineers and the United Engineering Society, which are now conjointly administered by a Library Board of the last-named Society, have on their shelves over 60,000 books on technical subjects, and they receive currently over 1,000 engineering or scientific periodicals.

For those members who are personally able to use the library a bibliography on any desired engineering subject will, on request, be prepared by the Bureau, and with adequate prior notice, the volumes relating thereto will be set apart for personal perusal.

We desire that all members of the Institute shall become better acquainted with the splendid library facilities they own, and with the advantages which can be secured by those who avail themselves of the aid furnished by its Service Bureau.

NOMINATIONS FOR OFFICERS

The co-operation of the members of the Institute is earnestly sought by the Committee on Nominations, recently appointed by the Board of Directors, in its work of formulating a ticket for officers and places falling vacant in the Board of Directors in February, 1916. The Committee must present the official ticket of nominations to the Board of Directors at its October meeting. Members are, therefore, urged to send promptly their nominations for officers and Directors in order that the Committee may have an opportunity to give them full consideration.

The officers to be elected at the annual meeting in February, 1916, are: One officer, known as Director and President. Two officers, known as Director and Vice-President. Five officers, known as Director.

The officers of the Institute whose terms expire are as follows:

President, William L. Saunders (not eligible for re-election).

Past President, Charles F. Rand.

Vice-Presidents, Thomas H. Leggett, District 0; Fred W. Denton, District 4.

Directors, John W. Finch, District 7; John H. Janeway, District 0; Edward P. Mathewson, District 5; Joseph W. Richards, District 2; George D. Barron, District 0.

Communications should be sent to the Chairman of the Committee on Nominations, Crocker Bldg., San Francisco, Cal.

COMMITTEE ON NOMINATIONS: F. W. BRADLEY, *Chairman*.

LOUIS S. CATES,
R. C. GEMMELL,

JAMES F. KEMP,
FRANK A. ROSS,

FRANK M. SMITH,
ARTHUR THACHER.

PERSONAL

(Members are urged to send in for this column any notes of interest concerning themselves or their fellow-members.)

Members and guests who registered at Institute headquarters during the period July 10 to Aug. 10, 1915:

William H. Rettie, Yonkers, N. Y.
A. J. Eveland, Pachuca, Mexico.
Chang Fu Wang, Shanghai, China.
F. H. Fovargue, Terlingua, Tex.
W. W. Elmer, Tuttle town, Cal.

Paul S. King, Wilmington, Del.
E. E. R. Tratman, Chicago, Ill.
Courtenay Dekalb, Tucson, Ariz.
Emil Gathmann, Baltimore, Md.

Henry S. Drinker, E. M., LL. D., has just completed 10 years' service as President of Lehigh University. A notable gathering of alumni, with their wives and daughters, numbering more than 1,000 persons, met in South Bethlehem on June 5. In accordance with the request of Dr. Drinker, no formal celebration was held, but, nevertheless, a spontaneous and enthusiastic expression of appreciation and affection was voiced by alumni, students and faculty, testifying to the regard felt for Dr. Drinker and extolling the progress made by the University under his guidance.

Thomas T. Read, for several years Associate Editor of the *Mining and Scientific Press*, has accepted a position with the Engels Copper Co., Engels, Cal.

M. S. Brandt has severed his connection with the Seneca Consolidated Gold Mines Co., Seneca, Cal., and has resumed work with the White Eagle Mining & Smelting Co., Sunshine, Colo. He has also opened an office for general practice.

Milo W. Krejci, Metallurgist of the Boston & Montana Reduction Works of the Anaconda Copper Mining Co., has been made Assistant Superintendent.

Earl S. Bardwell, who was General Foreman of the concentrator, has been appointed Metallurgist of the Boston & Montana Reduction works.

Theodore P. Holt has been appointed Superintendent of the Tintic Milling Co., Silver City, Utah.

Benjamin C. Rogers has accepted a position in the Department of Geology of the Detroit Copper Co., Morenci, Ariz.

Arthur Clark Terrill, who as field representative of the New York State Museum, collected and arranged the large mining exhibit of the Empire State at the Panama-Pacific International Exposition, has been appointed head of the department of Mining Engineering at the University of Kansas, Lawrence, with the title of Professor of Mining and Ore Dressing, beginning Sept. 1, 1915.

William B. Phillips, formerly Director of the Bureau of Economic Geology and Technology at the University of Texas, Austin, will assume the Presidency of the Colorado State School of Mines, Golden, Colo., on Sept. 1, 1915.

S. M. Marshall is now manager of the turbine, condenser, and pump departments of the Southwark Foundry & Machine Co., Philadelphia, Pa.

Irving R. Gard is now consulting engineer for the Columbia River Coal Dock Co., North Portland, Ore.

Luther B. Eames has accepted the position of Mill Superintendent of the Hollinger Gold Mines, Ltd., Timmins, Ont.

Robert Ammon has been appointed Assistant Mill Superintendent for the American Zinc Co., Mascot, Tenn.

Charles E. Locke was a guest of alumni of the Massachusetts Institute of Technology at a recent informal dinner in Denver, and gave an illustrated lecture on the new home of that Institute.

Lawrence Arthur Callaway was granted the degree of Doctor of Science by Harvard University at the recent commencement exercises. The subject of his thesis was The Constitution of Iron-Copper Mattes.

Carl F. Schaber has been appointed division engineer for the Tennessee Coal, Iron & Railroad Co., Bessemer, Ala.

Drury McM. Phillips has severed his connection with the Consumer's Lignite Co., Hoyt, Tex., and has accepted a position with the engineering division of the refinery department of the Texas Company at its Port Arthur Refinery.

Walter E. Burlingame announces that the business of E. E. Burlingame & Co., assayers and chemists, will be conducted under his name at 1736-38 Lawrence St., Denver, Colo.

Milton A. Allen has been appointed Manager of the Pearsite Company, Inc., Cannel City, Ky.

Harry P. Sweeny is General Manager of the Canadian Atlantic Coal Co., Ltd., Port Malcolm, Nova Scotia.

John Orth Liebig is now employed by the Balbach Smelting & Refining Co., Newark, N. J.

C. Edwin Nighman has been appointed shift boss at the Leonard mine of the Anaconda Copper Mining Co., Butte, Mont.

ENGINEERS AVAILABLE

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons introduced by members.)

Member, technical graduate, aged 36. Experienced copper metallurgist and mining geologist. Familiar also, through 15 years' broad practical experience, with mining, cyanidation, concentration (including oil flotation), metallurgical accounting, and mine and plant management. Best references. No. 237.

Member, graduate engineer with 10 years' experience in malleable, gray iron, and steel, as chemist, metallographist, and metallurgist. Researches and investigations by laboratory methods and under works conditions. No objection to some routine. No. 243.

Member, technical graduate, broad experience in coal, oil, and clay. Will go anywhere as mining geologist after Sept. 1, 1915. No. 244.

Member, technical graduate, aged 30, married, with over 10 years' experience as chemist, engineer on construction, superintendent and general superintendent of bituminous coal and coke plants. Speaks Spanish. References. No. 245.

Member, open for engagement as mine safety inspector or as superintendent of bituminous coal mine. No. 246.

Member, graduate mining engineer, with five years' experience in coal mining as mining engineer, construction engineer, pit boss, and superintendent, open for engagement. Best references. No. 247.

Member, aged 39 years, 20 years' underground experience gained in Canada, Washington, California, Nevada, Colorado, Mexico and Central America. Speaks Spanish. Assayer and surveyor. Good on maps. Years of experience as foreman and superintendent. Up-to-date on mining costs and methods. Practical. Wants position as general mine foreman or assistant to superintendent. At present in Canada. No. 248.

Member, technical graduate, aged 32, married, 12 years' practical experience in mining and metallurgy of gold and copper, desires change. Now with company operating continuous decantation cyanide process. Available Sept. 1. No. 249.

Research chemist, technical graduate, aged 30. Seven years' varied experience as metallurgical chemist in several industrial concerns engaged in the manufacture of special alloys, both ferrous and non-ferrous. For the past 3½ years chief chemist in charge of two busy laboratories for present employers. Best reasons for desiring change. Excellent references from present and past employers. Open for engagement after Oct. 1. Correspondence arranging for personal interview invited. No. 250.

Member, aged 49, married, 25 years' experience in hydraulic mining: Speciality, opening up new properties. Not a college graduate. At present general manager of mining company. Open for engagement after Nov. 1, as manager or superintendent. No. 251.

Member, technical graduate, 8 years' experience, three in metal and five in coal mine engineering. Have made a study of ventilation and efficiency. 32 years old, single and temperate. Desire a position as mine or ventilation engineer, superintendent or inspector of mines, will go anywhere. Can report immediately. No. 252.

Member, technical graduate, wide experience in western United States, Mexico, South America and the Orient. Organization and management of properties. Economic and metallurgical investigation. Estimates design and construction of plants. No. 253.

ADVANTAGES TO MEMBERS OF OUR ADVERTISING SECTION

It is our intention to make the advertising section of the *Bulletin* as comprehensive as possible, so that members can turn to it for complete information as to where to buy the best mining and metallurgical supplies at the lowest prices. No firms or supplies of dubious reputation are advertised in the *Bulletin*. If members, in buying any material, will refer to the advertisement of that material in the *Bulletin*, or if they do not find it advertised in the *Bulletin*, will, when ordering, ask the manufacturers why the goods were not advertised in the Institute publication, it will be of material benefit to all of us, because the chief reason why advertisers are unable to trace direct results is because purchasers do not mention the *Bulletin* when writing. Finally, if members, when in need of anything whatsoever, will write to the Institute headquarters, an effort will be made to send them a complete list of manufacturers able to supply the desired materials of good quality. The furnishing of these lists will be a benefit to our Advertising Department, as well as to the members. We need scarcely call attention to the advantages obtained by the members from the money which advertising brings in, because all this money is returned directly to the members in one form or another.

LIBRARY

AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS

AMERICAN SOCIETY OF MECHANICAL ENGINEERS

AMERICAN INSTITUTE OF MINING ENGINEERS

UNITED ENGINEERING SOCIETY

WILLIAM P. CUTTER, Librarian

The Library of the above-named Societies is open from 9 A.M. to 10 P.M. on all week-days, except holidays, from September 1 to June 30, and from 9 A.M. to 6 P.M. during July and August. The Library contains about 55,000 volumes, including sets of technical periodicals and the publications of scientific and technical societies.

Members of the Institute, with few exceptions, are forced to spend a portion of their time in localities isolated from sources of information. To these the Library can render valuable service through correspondence; letters requesting information will receive special attention. The Library is prepared to furnish references and copies of articles on mining and metallurgical subjects; to determine the existence of mining maps, and to furnish general information as to the geology and mineral resources of all countries.

All communications should be made as definite as possible so that the information received may be what is desired and not include collateral matter which may not be of interest. The time spent in searching for such collateral matter will be saved, and the information will be sent more promptly and in more usable shape.

The members of the Institute can be of service to the Library by forwarding copies of mining reports, maps privately issued, and similar material, which will be classified, indexed, and made available to other members. Suggestions for additions to the Library, either by purchase or personal solicitation as gifts, will be welcomed. It is hoped that members while in the city will use the Library freely, and assurance is given that most careful service will be rendered to them.

LIBRARY ACCESSIONS

PARTIAL LIST CLASSIFIED BY SUBJECTS

Mining, Metallurgy, and Chemistry

- ATLANTIC CITY GOLD MINING DISTRICT, FREMONT COUNTY. (Bull. 7, Wyoming State Geologist.) Cheyenne, 1914.
- CHEMISTRY OF PETROLEUM AND ITS SUBSTITUTES. A Practical Handbook. By C. K. Tinkler and F. Challenger. London, 1915.
- CHEMICAL ENGINEERING NOTES ON GRINDING, SIFTING, SEPARATING AND TRANSPORTING SOLIDS. By J. W. Hinchley. London, 1914.
- HANDBUCH DER MINERALCHEMIE. Bd. 11, pt. 7. By C. Doelter. Dresden, 1915.
- KUPFER. By W. Borchers. Halle, 1915.
- COKE-OVEN ACCIDENTS IN THE UNITED STATES DURING THE CALENDAR YEARS 1913, 1914. (Technical Paper 118, U. S. Bureau of Mines.) Washington, 1915.
- PRACTICAL COAL MINING. Ed. 5. By George L. Kerr. London, 1914.
- METHODS OF PREVENTING AND LIMITING EXPLOSIONS IN COAL MINES. (Technical Paper 84, U. S. Bureau of Mines.) Washington, 1915.

- OCCURRENCE OF EXPLOSIVE GASES IN COAL MINES.** (Bull. 72, U. S. Bureau of Mines.) Washington, 1915.
- PERMISSIBLE EXPLOSIVES, TESTED PRIOR TO MARCH 1, 1915.** (Technical Paper 100, U. S. Bureau of Mines.) Washington, 1915.
- WYOMING. MINING LAWS, FEDERAL AND STATE, 1913.** (Bull. 6, Wyoming State Geologist.) Cheyenne, 1913.
- THE TINTIC MINING DISTRICT OF UTAH,** prepared for the Tintic meeting of the Utah Section of the American Institute of Mining Engineers, July 15, 1915, with map. (Gift of Chief Consolidated Copper Co.)

Geology and Mineral Resources

- CLAY AND SHALE DEPOSITS OF THE PROVINCE OF QUEBEC, PRELIMINARY REPORT.** (Memoir 64, Canada Department of Mines.) Ottawa, 1915.
- COAL FIELDS OF BRITISH COLUMBIA.** (Memoir 69, Canada Department of Mines.) Ottawa, 1915.
- COAL FIELDS OF MANITOBA, SASKATCHEWAN, ALBERTA, AND EASTERN BRITISH COLUMBIA.** (Memoir 53, Canada Department of Mines.) Ottawa, 1914.
- LES MINÉRAIS DE FER DE LA PROVINCE DE QUÉBEC, GISEMENTS ET UTILISATION.** Quebec, 1915.
- MARBLE AND MARBLE WORKING.** By W. G. Renwick. London, 1909.
- BASIN AND GREYBULL OIL AND GAS FIELDS, BIGHORN COUNTY, WYOMING.** (Bull. 10, Wyoming State Geologist.) Cheyenne, 1915.
- PROSPECTIVE OIL FIELDS AT UPTON, WESTON COUNTY, ETC.** (Bull. 5, Wyoming State Geologist.) Cheyenne, 1913.
- SALT CREEK OIL FIELD, NATRONA COUNTY.** (Bull. 8, Wyoming State Geologist.) Cheyenne, 1914.
- LITTLE BUFFALO BASIN OIL AND GAS FIELD.** (Bull. 11, Wyoming State Geologist.) Cheyenne, 1915.
- GEOLOGY AND ORE DEPOSITS OF COPPER MOUNTAIN AND KASAAN PENINSULA, ALASKA.** (Professional Paper 87, U. S. Geological Survey.) Washington, 1915.

General

- CATALOGUE OF TECHNICAL PERIODICALS IN LIBRARIES IN THE CITY OF NEW YORK AND VICINITY.** Compiled by A. J. Gates. New York, 1915.
- EXPERIENCES IN EFFICIENCY.** By B. A. Franklin. New York, 1915.
- SCIENCE AND PRACTICE OF MANAGEMENT.** By A. H. Church. New York, 1914.
- GUIDE BOOK OF THE WESTERN UNITED STATES. PART B—THE OVERLAND ROUTE.** (Bull. 612, U. S. Geological Survey.) Washington, 1915.
- INTERNATIONAL MINING MANUAL, 1915.** Denver, 1915.
- POOR'S MANUAL OF PUBLIC UTILITIES, 1915.** New York, 1915.
- MODERN PUMPING AND HYDRAULIC MACHINERY.** By Edward Butler. London, 1913.
- STRAIGHT LINE ENGINEERING DIAGRAMS.** By Manifold and Poole. San Francisco, n.d.
- STRUCTURAL ENGINEERING.** By J. E. Kirkham. Chicago-London, 1914.
- STRUCTURAL ENGINEERS' HANDBOOK.** By M. S. Ketchum. New York, 1915.

Trade Catalogues

- BESSEMER GAS ENGINE Co.,** Grove City, Pa. Bessemer Monthly. July, 1915.
- TROPENAS CONVERTER Co.,** New York, N. Y. Steel Castings. Nos. 4, 5. Feb., July, 1915.

MEMBERSHIP

NEW MEMBERS

The following list comprises the names of those persons who became members during the period July 10 to Aug. 10, 1915:

Members

- ALBERTSON, MAURICE MERTON, Mine Surveyor, Mining Corporation of Canada, Cobalt, Ont., Canada.
- ARMSTRONG, GEORGE CHANDLER..... Mgr., Indian Refining Co., Lawrenceville, Ill.
- BARBER, MAX HAYDEN, Min. Engr..... Supt., Crosby Mine, Nashwauk, Minn.
- BARNESON, JOHN..... Merchant, 310 Sansome St., San Francisco, Cal.
- BENSEL, JOHN A., Cons. Engr..... 111 Broadway, New York, N. Y.
- BORGESON, ANSHELM CYRUS, Min. Engr..... Shenango Mine, Chisholm, Minn.
- BROOKS, CHARLES PETER, Min. Engr..... 228 Dooly Bldg., Salt Lake City, Utah.
- BROWN, FRANK HAROLD, Cons. Min. Engr., Federal Co., Inc., 50 Broad St., New York, N. Y.
- CLARK, DOUGLAS..... Kellogg, Idaho.
- CRESSON, BENJAMIN FRANKLIN, JR., Chief Engr., Dept. of Commerce and Navigation, Jersey City, N. J.
- CROSSFIELD, J. T. K., Min. Engr., Surveyor, Chile Exploration Co., Chuquicamata, Chile.
- DALY, WILLIAM BARRY, Genl. Supt. of Mines, Anaconda Copper Mining Co., 508 Hennessy Bldg., Butte, Mont.
- DUPUIS, OLIVER CAMILLE, Mining, Mandarin Mines Corp., Yellow Jacket, Idaho.
- EDWARDS, CHARLES LEWIS TAYLOR, Blast Furnace Assistant, Bethlehem Steel Works, So. Bethlehem, Pa.
- GRAFF, WILBUR W., Supt., North Lake District, Cleveland-Cliffs Iron Co., Ithpeming, Mich.
- GUGGENHEIM, DANIEL, Pres., American Smelting & Refining Co., 120 Broadway, New York, N. Y.
- HAMMON, WENDELL C., Mining..... 433 California St., San Francisco, Cal.
- HANDLEY, HARVEY LOCKHART..... Genl. Supt., Yankee Fuel Co., Yankee, N. M.
- HARLEY, GEORGE TOWNSEND, Experimental work, Arizona Copper Co., Ltd., Morenci, Ariz.
- HARRINGTON, DANIEL, Min. Engr..... U. S. Bureau of Mines, Pittsburgh, Pa.
- HERMAN, JOHN, Assayer and Chem..... 514 S. Los Angeles, Los Angeles, Cal.
- HILBY, GEORGE ROBERT, Engineering and contracting..... Salinas, Cal.
- HODGE, JOHN E., Min. Engr., Vice-Pres., E. J. Longyear Co., Minneapolis, Minn.
- HODGKINSON, HAROLD HOWE, Min. Engr., New Jersey Zinc Co., Franklin Furnace, N. J.
- HOLBROOK, E. A..... Asst. Prof. of Mining Engineering, Univ. of Illinois, Urbana, Ill.
- KOCH, ARTHUR WILLIAM, Mech. Draftsman, Care American Zinc Co. of Tenn., Mascot, Tenn.
- KWONG, WU..... Director, Tung Hsing Colliery, Men-tou-kou, via Peking, China.
- LAROQUE, FULGENCE, Min. Engr.; Shift Boss, Durban Roodepoort Deep Gold Min. Co., Ltd., Durban Deep, Roodepoort, Transvaal, So. Af.
- MACISAAC, FREDERICK J., Contractor..... 18 E. 41st St., New York, N. Y.
- MARSH, AUSTIN GERRY, Met., Concentrator Dept., Nevada Consolidated Copper Co., McGill, Nev.
- NELSON, CARL NELSON, Min. Engr.; Mine Foreman, New York & Honduras Rosario Min. Co., San Juancito, Honduras.
- PETTIS, EDSON S., Met.; Supt., California Ore Testing Co. Plant, San Francisco, Cal.
- REED, AVERY H., Min. Engr..... Cons. Min. Engr., Reed & Wilson, Marion, Ky.
- ROBERTS, HUGH M., Min. Geol..... 710 Security Bank Bldg., Minneapolis, Minn.
- ROBERTS, WILLIAM PAXTON, Mining; Asst. Mine Surveyor, Witherbee, Sherman & Co., Mineville, Essex Co., N. Y.
- SALMON, HERBERT S., Chief Mine Inspector, Ore Mines, Care Tennessee Coal, Iron & R. R. Co., Bessemer, Ala.
- SLAUGHTER, BEN GABRIEL, Chief Engr., Canadian Copper Co., Copper Cliff, Ont., Canada.

STACK, FRANK LAWRENCE, Min. Engr.....	1547 Euclid Ave., Berkeley, Cal.
STRACHAN, FRANCIS J., Supt., Concentrator, Cananea Cons. Copper Co.,	Cananea, Son., Mexico,
TALLY, ROBERT E.....	Supt. of Mines, United Verde Copper Co., Jerome, Ariz.
THOMPSON, LESTER S., Min. Engr.....	1373 Washington Ave., New York, N. Y.
TOBELMANN, HENRY ADOLPH, Met.; Chem., New Cornelia Copper Co., Ajo,	Pima Co., Ariz.
WANG, CHENG-FU.....	3 Quinsan Gardens, Shanghai, China.

Associate Members

ACKERMAN, ERNEST ROBINSON, Pres., Lawrence Portland Cement Co., Plainfield, N. J.	
ALBIN, LELAND DAY, Asst. Gen. Sales Mgr., Ingersoll-Rand Co., 11 Broadway,	New York, N. Y.
CONNABLE, FRANK L., Vice-Pres., E. I. du Pont de Nemours Powder Co.,	Wilmington, Del.
GODDARD, PETER LOUIS, Smelter Foreman, Mount Elliott, Ltd., Selwyn,	Cloncurry, Queensland, Australia.
LEITER, JOSEPH, Coal Operator.....	Dupont Circle, Washington, D. C.
PATTERSON JOHN H.....	Pres., National Cash Register Co., Dayton, Ohio.

Junior Members

CROSTON, JOHN J., Met.....	242 Church St., Grass Valley, Cal.
DAMMANN, ARTHUR CHESTER, Min. Engr.....	Fairview, Nev.
DAVID, JOSEPH, Student.....	233 College Ave., Houghton, Mich.
EMMERSON, ERIC ROY.....	Port Arthur, Ont., Canada.
HICKS, FRANK VIVYAN, Engr.....	Care General Delivery, Bisbee, Ariz.
KAANTA, HENRY W., Assayer and Chem., Cariman Min. & Mill. Co., Cardinal, Colo.	
ROHRER, WALTER ELMER, Asst. Engr....	Central Coal & Coke Co., Pittsburg, Kan.
SCHUYLER, ARENT HENRY.....	404 Riverside Drive, New York, N. Y.
TAYLOR, WILLIAM WALTER, Engr., Canadian Copper Co., Copper Cliff, Ont., Canada.	
WRIGHT, FREDERICK SYLVESTER, Asst. Chem., Boston & Montana Smelter,	Great Falls, Mont.

Change of Status—Junior Member to Member

PRATT, ALLYNE FRANCIS.....	727 E. 43d St. N., Portland, Ore.
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CANDIDATES FOR MEMBERSHIP

APPLICATION FOR MEMBERSHIP.—The Institute desires to extend its privileges to every person to whom it can be of service. On the other hand, it is not desirable that persons should be admitted to membership in classes for which they are not qualified. Members of the Institute can be of great service if they will make a practice of glancing through the list of applicants and promptly notifying the Committee on Membership, or the Secretary of the Institute, of any persons whom they think should not be classified in accordance with the list given.

The following persons have been proposed during the period July 10 to Aug. 10, 1915, for election as members of the Institute. Their names are published for the information of Members and Associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Board of Directors, which has the power of final election.

Members

Wesley Hudson Beck, Ellsworth, Pa.

Proposed by W. A. Luce, W. W. Keefer, Samuel A. Taylor.

Born 1867, Rome, N. Y. Prepared self for college. 1886-88, Civil Engineering, Lehigh Univ. 1888-90, Land Surveying, California. 1890-91, Transitman, Rome, Watertown and Ogdensburg R. R. 1891-1903, Transitman, Asst. Engr. Maintenance of Way, Resident Engr. Construction, New York Central & Hudson River R. R. 1903-10, Asst. Engr., Pittsburgh-Coal Co.

Present position: Asst. Chief Engr., Ellsworth Collieries Co.

James A. Block, Salt Lake City, Utah.

Proposed by J. M. Callow, Ernest Gayford, Walker J. Boudwin.

Born 1895, Salt Lake City, Utah. 1907-11, Salt Lake High School. 1911-14, University of Utah. 1915, Research work, U. S. Bureau of Mines, Salt Lake City. 1911, Draftsman, General Engrg. Co., 1913-15, Transitman and Draftsman, L. C. Stubbins, C. P. Brooks. 1914, Concrete inspector and engineer, Phoenix Construction Co.

Present position: Met. for J. M. Callow.

William M. Bradley, Salt Lake City, Utah.

Proposed by W. Fitch, J. R. Finlay, H. P. Henderson.

Born 1861, Elkhorn, Wis. 1883, Univ. of Wisconsin; LL. B. Owner of or interested in Centennial Eureka Min. Co.; Daly Judge Min. Co.; Silver King Cons. Min. Co.; Chief Cons. Min. Co.; Bennett Min. Co.; Centennial Emma Min. Co.; Good Hope Placer Min. Co., and other mines in the U. S. and Mexico.

Present position: Counsel for mining, smelting, railroad and other large corporate interests.

Grant David Bradshaw, Johnstown, Pa.

Proposed by D. M. Stackhouse, George E. Thackray, M. G. Moore.

Born 1884, Chicago, Ill. 1896-1901, Lewis Institute, Chicago, Ill. 1901-04, Univ. of Mich.; B. S. in M. E. 1904-09, Steam expert, draftsman, Secy. to Chief Engr., Foreman of Rod Mills, Joliet Works, Illinois Steel Co. 1909, Asst. Supt. Billet Mill, Indiana Steel Co., Gary, Ind. 1910, Consulting Engr., Chicago, Ill.

Present position: 1910 to date, Steam Engineer, Cambria Steel Co.

Ralph G. Bulkley, Montezuma, Colo.

Proposed by Frank Bulkley, Richard A. Parker, S. A. Ionides.

Born 1892, Aspen, Colo. 1909, Grad., East Denver High School. 1911, Phillips Academy, Andover, Mass. 1914, Yale University, Sheffield Scientific School; Ph. B. 1914, Miner, Golden Cycle Mine, Victor, Colo. 1914, Concentrating mill, Portland Cons. Gold Min. Co., Victor, Colo. 1915, Min. Engr., Crested Butte Coal Co. and Crested Butte Anthracite Min. Co., Crested Butte, Colo.

Present position: Supt. of Concentrating Mill, Montezuma Mines and Mills Co.

Stanley C. Bullock, Collahuasi, Chile.

Proposed by J. H. Ivey, Frederick Hellmann, Henry Hay.

Born 1881, Aylsham, Norfolk, England. 1889-94, Private school. 1904-08, Lancing College, Shoreham, Sussex, England. 1898-1902, Pupil, Fraser & Chalmers Works, Erith, erecting, turning, fitting, etc. Four years Engrg. course, Woolwich Technical College. 1902-05, Drawing office; mining, milling, smelting machinery and plants; details and design. Three years, assaying course, Woolwich Technical College. 1905-07, Draftsman and Asst. Engr., Bewick, Moreing & Co., London. Assaying and metallurgy course, Birkbeck College. 1907-09, Asst. Engr. and draftsman, Bewick, Moreing & Co., Australia. Australian Smelting Corporation. Surveying course, Melbourne School of Mines. 1909-11, Constructional Engr. Junction North Mine, Broken Hill (N.S.W.). Designed and erected new lead concentration and flotation plants; full charge during Chief Engineer's absence. 1911, Reporting on lead-zinc property, Norway. Sampled mines and advised on reconstruction of mill. 1911-12, Asst. Engr., Spassky Smelter Siberia; had charge of blast furnaces, converters and refinery. Learnt Russian and Kirghese. 1912-13, Reporting for various London firms. 1913-15, Mine and Mill Supt., Société des Mine de Cuivre, Collahuasi, La Grande, Chile.

Present position: Genl. Mgr., Poderosa Min. Co.

Albert Cameron Burrage, Jr., Boston, Mass.

Proposed by Huntington Adams, Fred Hellmann, E. A. C. Smith.

Born 1887, Boston, Mass. 1904-08, Harvard; B. A. 1908-10, C. S. Bradley & Sons, metallurgical work. 1910-12, Copper Extraction Co., metallurgical work.

Present position: 1912 to date, Director, Chile Copper Co.

J. H. Champion, Leadville, Colo.

Proposed by David G. Miller, W. W. Davis, William J. Cox.

Born 1856, Cornwall, England. 1874, Attended South Kensington class of Arts and Sciences; circumstances alone prevented pursuance of these studies. 1876, Contracted with Brendon Hills mine, Somersetshire, to produce iron ore. 1877-81, Contract work for Terrible Min. Co., Georgetown, Colo. 1881-86, Foreman, Small Hopes Co., Leadville, Colo. 1886-1890, Operating lease on Morning Star mine, Leadville, Colo. 1890-95, General contract work—sinking shafts and driving tunnels, Leadville, Colo. 1895-96, Foreman, Forest Queen mine, Leadville, Colo. 1896-1901, Foreman, The Yak Min., Mill. & Tunnel Co., Leadville, Colo.

Present position: 1901 to date, Supt., Yak Min., Mill. & Tunnel Co.

William Andrews Clark, Jr., Butte, Mont.

Proposed by W. A. Clark, Horace V. Winchell, Fred T. Greene.

Born 1877, Deer Lodge, Mont. 1899, University of Virginia; B. L.

Present position: Pres., Elm Orlu Min. Co., and Timber Butte Mill. Co.

Grant H. Dowell, Douglas, Ariz.

Proposed by C. Legrand, P. P. Butler, G. D. Van Arsdale.

Born 1866, Lexington, Ill. Up to 1884, common and high school, Kansas. 1892, Course, Kansas City Business Univ. 1897-1904, Chief Clerk and Asst. to Mgr., Cons. Kansas City Smelt. & Refg. Co., El Paso, Tex. 1904-10, Met. Accountant and ore buyer, Copper Queen Cons. Min. Co., Douglas, Ariz. 1910-12, Supt., Old Dominion Copper Min. & Smelt. Co., Globe, Ariz.

Present position: 1912 to date, Asst. Genl. Mgr., Copper Queen Cons. Min. Co.

Ensor R. Dunsford, Quarry, N. S., Canada.

Proposed by H. J. Brown, Laurance I. Neale, F. C. Fearing.

Born 1887, Lindsay, Ont., Canada. Grammar and high schools, Columbus, O. 1911, Ohio State Univ.; E. M. 1911, Asst. Engr., Smithers Creek Dist., Sunday Creek Coal Co., Longacre, W. Va. 1912, Min. Engr., Darby Coal Min. Co., Darbyville, Va. 1912-15, Salesman and demonstrator, Myers-Whaley Co., Knoxville, Tenn.

Present position: Asst. Supt., Victoria Gypsum Min. & Mfg. Co., Ltd.

G. Marshall Gillette, Jenkins, Ky.

Proposed by Newell G. Alford, Howard N. Eavenson, Edward O'Toole.

Born 1879, Cumberland, Md. Prior to 1897, common school, Philadelphia, Pa., and Cumberland, Md. 1897-99, Princeton Univ.; B. S. 1903-13, Supt., Penn. Div.; 1913, Asst. Genl. Supt., Elk Horn Div.; 1913-14, Mgr., Millers Creek Div., Consolidation Coal Co., Van Lear, Ky.

Present position: Mgr., Elk Horn Div., Consolidation Coal Co.

Dorsey Hager, Tulsa, Okla.

Proposed by H. A. Wheeler, P. N. Moore, Arthur Thacher.

Born 1887, St. Paul, Minn. 1902-05, St. Louis Manual Training School. 1905-06 and 1907-09, Washington Univ., St. Louis, Mo. 1909-10, Columbia Univ., New York; specialized in Geology. Studied science and engineering at Washington Univ. 1906-07, Engrg. dept., St. Louis Street Railways. 1907, Katherine mine (lead), Fredericktown, Mo. 1909, Coal prospecting, Hamilton, Ia. 1910-12, Southern Pacific oil dept. (K. T. & O. Co.), San Joaquin Valley oil fields, Cal. Two and one half years, engrg. and geol. work, Cal. oil fields. Eight months, geol. work, Okla. oil fields. Also examinations made in N. M. and La.

Present position: Cons. work, Petroleum Geol. and Engr.

Oliver Hall, Coniston, Ont., Canada.

Proposed by C. V. Corless, John F. Robertson, R. W. Schultz.

Born 1879, Washington, Ont., Canada. 1903, Grad., McGill Univ.; E. M.; 1904, M. Sc. 1900-04, Mining, railway work (Grand Trunk) and Allis-Chalmers, Chicago (summers only). 1904-05, Engr., West Canadian Collieries, Blairmore, Alta., Canada.

Present position: 1905 to date, Mines Supt., Mond Nickel Co.

Ralph Lucius Healy, Thane, Alaska.

Proposed by B. L. Thane, Angus Mackay, B. B. Nieding.

Born 1887, Marshall, Minn. 1905, High school, Red Lake Falls, Minn. 1906-08, Univ. of Minnesota, College Elec. Engrg. 1909-11, Univ. of North Dakota, College Min. Engrg.; E. M. 1911-12, Vesuvius Mines Co., Bohemia, Ore. 1912-13, Northern Ontario Exploration Co., Juneau, Alaska.

Present position: 1913 to date, Asst. Engr., Perserverance Mine, Alaska Gastineau Min. Co.

Herbert Benton Henegar, Mascot, Tenn.

Proposed by Robert Ammon, M. H. Newman, H. K. Sherry.

Born 1891, Knoxville, Tenn. 1898-1908, Knoxville City School. Diploma from High School. 1908-12, Univ. of Tennessee; B. S. 1907-12, Summer vacations, surveying. 1912-15, drill runner, mine and mill sampler, warehouse man, surface foreman, American Zinc Co. of Tenn.

Present position: Asst. Mine Foreman, American Zinc Co. of Tenn.

Edwin Gilbert Izod, Johannesburg, So. Africa.

Proposed by W. L. Saunders, George A. Howells, R. S. Carter.

Born 1876, Southsea, England. 1885-93, Vickers Army and Navy College, Southsea, England. 1893-99, Pupil, Maudsley Sons & Field, Lambeth, London (marine engineers). 1899-1901, University College, London; Junior and Senior Technical Course. 1901-10, Williams & Robinson, Ltd., Rugby, from experimental asst. to Asst. Managing Director.

Present position: 1910 to date, Cons. Elec. and Mech. Engr., Central Min. & Investment Corp., Ltd., and Rand Mines.

William Ralph Knapenberger, Mascot, Tenn.

Proposed by A. W. Koch, J. N. Houser, Robert Ammon.

Born 1890, Brunswick, Mo. 1907, Brunswick Public School. 1908, St. Charles Military Academy. 1913, Missouri School of Mines; B. S. 1912, Asst. Chem., National Malleable Castings Co., Toledo, O. 1913-14, Investigator, Anaconda Copper Min. Co., B. & M. Reduction Wks., Great Falls, Mont.

Present position: Assistant in Mill, American Zinc Co.

Grover B. Lantz, San Francisco, Cal.

Proposed by G. H. Clevenger, Robert H. Bradford, H. W. Young.

Born 1889, Milwaukee, Wis. 1895-1907, Grammar and High School, Colton, Cal. 1908-13, Stanford Univ.; A. B. in Min. and Met. 1909, Naval Mill Co., Guanajuato, Mex. 1909, Guanajuato Amalgamated Mines Co. 1913, Eagle Shawmut Mines Co., Shawmut, Cal. 1913, Nevada Mill Co., Tonopah, Nev. 1914, Colorado Min. Co., Arroyo, Mashate, P. I.

Present position: Asst. to A. D. Marriott, Jr., Dorr Cyanide Machinery Co.

Bernard Hugo van der Linden, San Francisco, Cal.

Proposed by Robert B. Moran, W. A. Williams, W. R. Hamilton.

Born 1880, Buitenzorg, Java. 1905, Univ. of Delft; M. E. 1906-07, Asst. in Petrography, Crystallography and Mineralogy, Univ. of Delft, Holland. 1907-13, Geol., Royal Dutch Oil Co., D. E. I.

Present position: 1913 to date, Chief Geol. and Field Mgr., Shell Company of California (California Oilfields, Ltd.).

Bruce R. Lindsay, Smithfield, Fayette Co., Pa.

Proposed by M. G. Moore, D. M. Stackhouse, Andrew B. Crichton.

Born 1883, Plymouth, Pa. 1890-1901, Plymouth Public Schools, Pa. Have first-grade mine-foreman certificate for bituminous mines of Pa. Mine-rescue certificate, Bureau of Mines. 1901-06, Employed on Engrg. Corps, Parrish Coal Co., Plymouth, Pa. 1906-08, Res. Engr., New River Collieries Co., Eccles, W. Va., in charge of mine work and shaft sinking. 1909, Dist. Engr., Lehigh Coal & Navigation Co., Lansford, Pa. 1910-15, Three years, Engr.; two years, Mine Supt., Cambria Steel Co., Johnstown, Pa.

Present position: Mine Supt., Republic Iron & Steel Co., Youngstown, O.

Hugh B. Lowden, Denver, Colo.

Proposed by S. A. Ionides, Richard A. Parker, J. W. Richards.

Born 1872, New York, N. Y. 1880-89, Public and private schools, and subsequently studied mathematics, chemistry, metallurgy, etc., privately. No degrees. Member, Colorado Scientific Society. 1890, Entered employ of Gates Iron Works, Chicago, Ill., manufacturers of rock and ore crushing, milling, and smelting equipment. 1905, Colorado Iron Works Co., Denver, Colo., metallurgy of the precious metals, copper and lead.

Present position: Secy., Colorado Iron Works Co.

George Caldwell McFadden, Witt, Ill.

Proposed by H. H. Stoek, E. A. Holbrook, H. E. Smith.

Born 1888, Lessmahage, Lanarkshire, Scotland. 1893-1901, Grade school, Danville, Ill. 1901-05, High school, Danville, Ill. 1905-08, Asst. Engr.; 1908-10, Div. Engr., Dering Coal Co., Westville, Ill. 1910-11, Div. Engr., Standard Collieries Co., Johnston City, Ill. 1911-12, Div. Engr., Burnwell Coal Co., Witt, Ill. 1912-14, Div. Engr., Peabody Coal Co., Witt, Ill.

Present position: 1914 to date, Div. Engr., Receivers, C. & E. I. coal properties.

Henry Douglas MacDonald, Mascot, Tenn.

Proposed by A. W. Koch, J. N. Houser, Robert Ammon.

Born 1887, Ludlow, Vt. 1907-08, Tufts College. 1908-13, Mass. Inst. of Tech., B. S. 1911, Min., Eustis Min. Co., Eustis, Que. 1913-14, Sampling, refining, diamond drill, Hollinger Gold Mines, Timmins, Ont.

Present position: 1914 to date, Asst. Met., American Zinc Co.

George Alexander Macready, Brighton, Trinidad, B. W. I.

Proposed by G. H. Clevenger, H. W. Young, D. M. Folsom.

Born 1885, Tacoma, Wash. 1906, Grad., Throop Polytechnic Institute, Pasadena, Cal. 1910, Grad., Leland Stanford Junior Univ.; A. B. in Geol. and Min. 1914, Grad., student in Geol., Univ. of Chicago. 1910-11, Resident Geol., North American Oil Consolidated, Taft, Cal. 1911, Instr., field and topographic Geol., Leland Stanford Junior Univ. 1911-14, Head Field Geol., Bermudez Co., and Caribbean Petroleum Co.; Venezuela.

Present position: 1914 to date, Resident Geol., Trinidad Lake Petroleum Co., Ltd.

W. Fred Nance, New Castle, Tex.

Proposed by George B. Rodgers, O. M. Bilharz, E. J. Rossbach.

Born 1884, Newnan, Ga. Grad., high school. Three years college course. Knowledge of engrg. acquired through experience and correspondence courses.

Present position: 1908 to date, Sec., Treas., Mine Supt. and Min. Engr., Belknap Coal Co.

Thomas Henry Oxnam, Jr., Fairview, Ariz.

Proposed by Philip Wiseman, Seeley W. Mudd, Frank A. Keith.

Born 1883, Lebanon, Ill. Preparatory school work, Los Angeles public schools, high school and Belmont Military Academy. 1905-06, Special work, Columbia Univ. 1897-98, Asst. Assayer, De Lamar Mines Co., De Lamar, Nev. 1901, 1903-05, Assayer and later Asst. Mine Supt., Palmarejo & Mexican Gold Fields, Mex. 1907-08, Chem., later Supt., for Receiver, Ariz. Smelting Co., Humboldt, Ariz. 1909, Underground miner, Copper Queen Mines, Bisbee, Ariz. 1909-10, General mill work, Goldfield Cons. and Combination Fraction, Goldfield. 1911, In charge of Min. Dept., Y. M. C. A. Technical School, Los Angeles, Cal. 1912-13, Met. Engr., Chino Copper Co., Hurley, N. M. 1914, Mine Supt., Gold Mountain Mines Co., Doble, Victorville, San Bernardino Co., Cal.

Present position: Met. Engr., Grand Reef Mill. Co.

Robert Dun Patterson, Lowe, W. Va.

Proposed by John J. Lincoln, L. E. Tierney, William D. Ord.

Born 1881, Dayton, O. 1904, Grad., Sheffield Scientific School, Yale University; Ph.B. Prepared for college at Taft School, Watertown, Conn. 1904, Came to Pocahontas coal field and assisted engr. of Weyanoke C. & C. Co., which place was just opening up. Later became Supt. 1912, Gen. Supt., Weyanoke C. & C. Co., White Star Min. Co., and Gulf Coal Co.

Present position: Genl. Mgr., Weyanoke White Star and S. J. Patterson Poca. Co.

John Painter Remick, Peking, China.

Proposed by Thomas T. Read, B. B. Thayer, J. W. Allen.

Born 1867, Ruabon, North Wales. To 1885, Various schools in England. 1885-87, Owen's College, Manchester. Student to J. & P. Higson, five years, out of which three years were spent in practical training at Hemsworth Collieries, Yorkshire and several collieries in Lancashire, Staffordshire, Cheshire, and North Wales. 1890-93, Asst., J. & P. Higson, Manchester. 1893-1907, Cons. Min. Engr. In charge of Flimby and Broughton Moor Collieries, Cumberland, Hockly Hall and Whalday Collieries, Warwickshire, etc. Certificated Colliery Manager, 1st class, a member of the I. M. E. (British), Iron and Steel Institute, Institution Min. and Met. A. M. I. C. E. Present position: General Agent & Engr., Pekin Syndicate, Ltd.

Charles Albert Schieren, New York, N. Y.

Proposed by Theodore L. Dwight, W. R. Ingalls, Richard H. Vail, W. L. Saunders, Frank W. Iredell, Leon P. Feustman.

Born 1869, Brooklyn, N. Y. 1879-1887, Polytechnic Institute of Brooklyn, N. Y. Traveled and founded agencies and branch houses in Europe, Cuba, Mexico, and all parts of U. S. A.

Present position: Pres., Charles A. Schieren Co.

Thomas McCourt Smither, Wonder, Nev.

Proposed by E. E. Carpenter, H. M. Alley, George J. Young.

Born 1892, Temple, Tex. 1900-05, Trinity School, New York, N. Y. 1905-06, Lowell High School, San Francisco, Cal. 1906-09, Univ. of Nevada. Preparatory School, Reno, Nev. 1909-13, Mackay School of Mines, Univ. of Nev., Reno, Nev.; B. S. in Min. Engrg. 1913-14, Montana-Tonopah Min. Co., Tonopah, Nev.

Present position: 1914 to date, Asst. Min. Engr., Nevada Wonder Min. Co.

William Leonard Stevens, Rancagua, Chile.

Proposed by Ross E. Douglass, Lester E. Grant, B. T. Colley.

Born 1878, Somerville, Mass. To 1896, Somerville, Mass., Public School. 1896-1900, Mass. Inst. of Tech.; S. B. 1900-01, Asst., Min. Dept., Mass. Inst. of Tech. 1901-05, Asst. Chem., Furnace Foreman, Genl. Foreman and Asst. Supt., Tenn. Copper Co. 1905-08, Supt., Tenn. Copper Co. 1908-10, Supt., Smelter, Copiapo Min. Co., Copiapo, Chile.

Present position: 1910 to date, Met. Accountant, Braden Copper Co.

John L. Templemen, Butte, Mont.

Proposed by W. D. Mangam, Horace V. Winchell, Fred T. Greene.

Born 1872, England. 1897, Iowa College, Grinnell, Ia.; A. B. 1899, Univ. of Virginia, Charlottesville, Va.; LL.B. 1915, Owner of and operating a lease of Nancy Hanks mine, Garnet, Mont. Owning and operating other properties in Montana of less consequence.

Present position: Attorney at law, Butte, Mont.

Arthur Thomas Ward, Sewell, Chile.

Proposed by B. T. Colley, R. K. Stockwell, Lester E. Grant.

Born 1890, Bellefonte, Pa. Until 1908, public and private schools, Bellefonte and Carlisle, Pa. Grad., Electrometallurgy, Lehigh Univ. Summers, 1908 to 1912, inc., sales dept., Aluminum Cooking Utensil Co., Phila. and New York Dist. 1913-14, Experimental Chem., U. S. Metals Refg. Co., Chrome, N. J.

Present position: 1914 to date, Experimental Engr., Braden Copper Co.

Francis Watson, Iquique, Chile.

Proposed by Huntington Adams, R. M. Atwater, Jr., C. R. Corning.

Born 1884, Lima, Peru. 1894-98, Barham House, a private school at St. Leonards-on-sea, England. 1898-1900, Harrow School, England. 1900-01, Junior clerk, Thomson Aikman, Jr., nitrate of soda broker, London, E. C. 1901-03, Unauthorized clerk, Edward Lawrence & Co., London Stock Exchange. 1903-09, Clarke, Bennett & Co. of Iquique, Chile, employed in nitrate manufactories. 1909-10, Nitrate Agencies, Ltd., successors of Clarke, Bennett & Co., as manager of "Oficina" Santa Fè, Tocopilla.

Present position: Nitrate Agencies, Ltd., as Mgr., Nitrate Department, controlling 15 nitrate manufactories, and all exploration work.

Arthur Edward Wells, Berkeley, Cal.

Proposed by F. G. Cottrell, Robert H. Bradford, G. H. Clevenger.

Born 1884, Saxonville, Mass. 1890-1902, Public Schools, Framingham, Mass. 1902-06, Mass. Inst. of Tech.; B. S. in Min. Engrg. and Met. 1906-13, American Smelt. & Ref. Co. 1906, Chem., Globe & Arkansas Valley Smelters. 1907, Chem., Murray Smelter. 1908-09, Research Chem., Murray and Garfield Smelters. 1910-11, Asst. Met., Garfield Smelter. 1912-13, Met., Garfield Smelter. 1914, Met. in charge investigation, Selby Smelter Commission.

Present position: Met., Smelter Smoke Investigations, U. S. Bureau of Mines.

Frederick H. Willcox, Pittsburgh, Pa.

Proposed by Van H. Manning, A. H. Fay, W. A. Williams.

Born 1882, Smyrna, N. Y. 1902, Colgate Academy. 1906, Mass. Inst. of Tech.; B. S. 1906-07, Asst. Instr., Mass. Inst. of Tech. 1907-11, Duquesne Furnace, Carnegie Steel Co. 1911-13, National Furnaces, National Tube Co. 1913-14, Monessen Furnaces, Pittsburgh Steel Co. 1914, U. S. Bureau of Mines, Pittsburgh, Pa.

Present position: Met. Engr., U. S. Bureau of Mines.

William Henry Williams, New York, N. Y.

Proposed by Eli T. Conner, E. R. Pettebone, C. Dorrance, Jr.

Born 1874, Athens, O. Public school, Toledo, O. Business college, Beaver Falls, Pa. 1891-92, Engrg. Corps, P. & L. E. R. R. 1892-1901, Various positions, Penn. Lines W. of Pittsburgh. 1901-1904, Asst. Secy., Asst. to Genl. Mgr., B. & O. R. R. 1904, Supt. Frt. Trans., Frisco and C. & E. I. 1905-07, Traffic Mgr., Pittsburgh Chamber of Commerce. 1907, Asst. to Pres., Delaware & Hudson Co.

Present position: 1907 to date, Vice-Pres., Delaware & Hudson Co., and Hudson Coal Co.

Associate Members

Reuben Waldo Davis, Dolores, Webb Co., Tex.

Proposed by J. F. Kemp, Bradley Stoughton, M. C. Scheble, E. R. Jones.

Born 1877, Cuyahoga Falls, O. 1883-85, Public school, Cuyahoga Falls, O. 1885-87, Public school, Anson, Tex. 1887-92 Public school, Santo Tomas, Webb Co., Tex. 1892-95, International Correspondence Schools, Scranton, Pa. 1902-03, New York University, New York. 1903-04, Columbia University, New York. Attended several other schools for short periods between 1902 and 1905. 1894-95, Santo Tomas Coal Co. 1895-96, Rio Grande Coal Co. 1896-1902, 1905-07, Cannel Coal Co. 1907-09, Cia. Fundidora de Fierro y Acero, Cia. Carbonifera del Norte.

Present position: 1909 to date; Engr., Maintenance of Way, Rio Grande & Eagle Pass Ry.; Asst. Supt. and Min. Engr., Cannel Coal Co.

Paul Messer, Tokyo, Japan.

Proposed by W. L. Saunders, George A. Howells, L. D. Albin.

Born 1873, Chicago, Ill. 1878-87, Oakland public and high schools, Chicago, Ill. 1887-90, Chicago Manual Training School. 1890-94, Cornell Univ., M. E. Member, Sigma Xi Society, A. S. M. E., Japan Inst. of Min. Engrs. 1894-96, Asst. and Chief Engr., Fraser & Chalmers Co. 1896-97, Foreman, machine shop, Myrpacht, Randfontein Mine, So. Africa. Amalgamator, North Randfontein Mine. 1897-1903, Shop work, Fraser & Chalmers, erecting floor. Engrg. Salesman, Fraser & Chalmers and Allis-Chalmers Co., Chicago, Butte, Rossland, Spokane, New York.

Present position: 1903 to date, Mgr., Engrg. Dept., and Acting Agt., American Trading Co.

Philip Edward Thomas, Johnstown, Pa.

Proposed by M. G. Moore, D. M. Stackhouse, Andrew B. Crichton.

Born 1892, Johnstown, Pa. To 1911, Public schools, Johnstown, Pa. 1915, Princeton Univ.; B. S. in Geol.

Present position: Coal Sales Agent, Shade Creek Coal Co.

Felix Richard John Wanjura, Koskullskulls, Gellivare, Sweden.

Proposed by W. L. Saunders, George A. Howells, L. D. Albin.

Born 1870, Beuthen, Germany. 1880-89, Uppsala High School. 1889-92, Stockholm Technic High School. 1893-99, Verein Königs and Laurahütte-coal-mines, Laurahütte, Germany.

Present position: 1899 to date; Bergv. aht. W. C. Freja, Koskullskulls Iron Mine.

Junior Members

Thomas Edwin Nott, Pittsburgh, Pa.

Proposed by Roswell H. Johnson, H. B. Meller, S. L. Goodale.

Born 1892, Spartanburg, S. C. 1908, Grad., Gainesville, Ga., High School.
1908-10, Univ. of Georgia, Athens, Ga. 1910-11, Univ. of S. C., Columbia, S. C.
1911-13, Clinchfield Fuel Co., Spartanburg, S. C. 1913, C. C. & O. Ry., Erwin,
Tenn. 1913, Clinchfield Coal Corp., Dante, Va.; 1913-14, Spartanburg, S. C.

Present position: Student, Univ. of Pittsburgh.

CHANGES OF ADDRESS OF MEMBERS

The following changes of address of members have been received at the Secretary's office during the period July 10 to Aug. 10, 1915. This list, together with the list published in *Bulletin* Nos. 100 to 104, April to August, 1915, and the foregoing list of new members, therefore, supplements the annual list of members corrected to Mar. 1, 1915, and brings it up to the date of August 10, 1915.

ALLEN, JOHN A.	Castano Nuevo, San Juan, Argentine.
ALLEN, MILTON A., Mgr.	Pearsite Co., Inc., Cannel City, Ky.
ALLPORT, ROBESON H.	724 W. Marshall St., San Antonio, Tex.
AMMON, ROBERT, Asst. Mill Supt.	American Zinc Co. of Tenn., Mascot, Tenn.
ARMS, CHARLES S.	5125 Wayne Ave., Philadelphia, Pa.
BASSETT, Thomas E.	Care Y. M. C. A., Bisbee, Ariz.
BATCHELLER, J. H., Supt., Ravenswood Mine,	U. S. Smelting Co., Reeds,
	Jasper Co., Mo.
BEHRE, HENRY A.	Care Braden Copper Co., Rancagua, Chile.
BELL, J. M., Geol. and Min. Engr.; Cons. Engr.,	L. Ehrlich & Co., 310 Dominion
	Express Bldg., Montreal, Canada.
BELLINGER, H. C., Care M. Guggenheim Sons,	Room 3546, 120 Broadway,
	New York, N. Y.
BENHAM, WILLARD MILES.	Ray, Ariz.
BERNHARD, DURWARD I.	Managua, Nicaragua.
BERRAY, NILES S.	349 Carroll Park E., Long Beach, Cal.
BLACKMAR, FRANK H.	706 Garfield Ave., Kansas City, Kan.
BLAKE, TRUE WALTER, Min. Engr.	Davis Creek Coal Co., Novinger, Mo.
BOISE, CHARLES WATSON, Chief of Mines de la	Forminière, Kasai, Tshikapa,
	Belgian Congo.
BOLER, CHARLES A.	135 S. Second St., Philadelphia, Pa.
BRADDOCK, HOMER F.	Mt. Pleasant, Pa.
BRANDT, MAURICE SHELDON, Chief Engr., White	Eagle Mining & Milling Co.,
	Boulder, Colo.
BRAYER, N. G., Supt., Sharon Works, National	Malleable Castings Co., Sharon, Pa.
CARPENTER, CLARK B.	Ray, Ariz.
CARSON, JOSEPH A.	Care Allis-Chalmers Mfg. Co., Milwaukee, Wis.
CARTHERS, A. WAYNE.	Old Company's Club, Lansford, Pa.
CHURCH, JOHN L.	Care Placer Hotel, Helena, Mont.
CRANDALL, RODERIC.	454 Forest Ave., Palo Alto, Cal.
DAKE, WALTER M., Jr., Met.	815 Gaylord St., Denver, Colo.
DEFTY, WILLIAM E.	618 N. Third Ave., Phoenix, Ariz.
DICK-CLELAND, A. F., Care Institution of Mining	and Metallurgy, 1 Finsbury
	Circus, London, E. C., England.
DIXON, A. FAISON, Field Mgr., Colon Development	Co., Ltd., Camp Rio de Oro,
	Venezuela.
EAMES, LUTHER B., Mill Supt., Hollinger Gold	Mines, Ltd., Box 228, Timmins,
	Ont., Canada.
ENOS, HERBERT C.	Carrizozo, N. M.
EVANS, CADWALLADER, JR.	R. F. D. 1, Sawtelle, Cal.
EVELAND, ARTHUR J.	Care Engineers' Club, 32 W. 40th St., New York, N. Y.
FORD, HAROLD PERCY.	2023 S. 20th St., St. Joseph, Mo.
FULTON, CHESTER A.	798 N. Broadway, Yonkers, N. Y.
GARD, IRVING R., Consulting Engr., Columbia	River Coal Dock Co.,
	North Portland, Ore.

- GILLESPIE, F. M. S. Abangarez, Costa Rica.
 GOODLAND, GILLMORE Devonshire Club, St. James St., London, England.
 GRANT, BERTRAM Kingman, Ariz.
 GREENAN, JAMES OWEN, Min. Engr.; Engr., Sarita Mining Co., Masonic, Cal.,
 via Sweetwater, Nev.
 GREYER, WALTER SCOTT, Min. Engr.; Asst. Min. Supt., Butte & Superior
 Copper Co., Ltd., Butte, Mont.
 GRIGGS, THOMAS W., Director, Colfax Consolidated Coal Co., Davenport, Iowa.
 HANCE, JAMES H. 202 Perry St., Elgin, Ill.
 HARBACH, HERBERT M. Care Lackawanna Club, Lackawanna, N. Y.
 HARDEN, EDWARD B. Panama Canal, Washington, D. C.
 HARDER, EDMUND CECIL, Geol. U. S. Geological Survey, Washington, D. C.
 HARDY, J. GORDON, American Smelting & Refining Co., 120 Broadway,
 New York, N. Y.
 HESS, RUSH M. Care Kenneth M. Hess, Coconut Grove, Fla.
 HOGBOOM, WILLIAM CORYELL, Drill Inspector, Southeast Missouri Lead Co.,
 Potosi, Washington Co., Mo.
 HOLT, THEODORE P. Supt., Tintic Milling Co., Silver City, Utah.
 HONNOLD, WILLIAM L. 5 London Wall Bldgs. London, E. C., England.
 HOTCHKIN, MERRITT W. Box 356, Haileybury, Ont., Canada.
 HOWARD, LESLIE E. 259 Niagara St., Lockport, N. Y.
 HOWAT, ANDREW M., Min. Engr.; Supt. of Mine, Mogollon Mines Co.,
 Mogollon, Socorro Co., N. M.
 HUDSON, ALBERT W., Met. Phelps, Dodge & Co., Douglas, Ariz.
 HUTCHINSON, ROBERT M. Care Lackawanna Club, Lackawanna, N. Y.
 INGLIS, J. F. Care B. & H. Mine, Twin Bridges, Mont.
 JENKS, NORMAN L. Care Tracey Bros., Medellin, Colombia, S. A.
 JENKS, THOMAS H. Box 173, Albuquerque, N. M.
 JOHNSTON, C. W. 122 Florida Ave., Port Norfolk, Va.
 KENNEDY, JOSEPH E. 120 Broadway, New York, N. Y.
 KIDDER, S. J., Genl. Mgr. Mogollon Mines Co., Mogollon, N. M.
 KING, PAUL S. 701 W. Tenth St., Wilmington, Del.
 KOEHLER, CARL F. Youngstown Sheet & Tube Co., Youngstown, Ohio.
 KONG, S. T. The "Riverside," Tungshan, Canton, China.
 LIEBIG, JOHN ORTH. Balbach Smelt. & Refining Co., Newark, N. J.
 LOERFABEL, W. HARRISON. 116 Ellison St., Lead, S. D.
 McKENNA, ALEXANDER G., Chem. and Met., Washington Steel & Ordnance Co.,
 Washington, D. C.
 McMEEKIN, CHARLES W. 1226 47th Ave., San Francisco, Cal.
 MACFARLANE, A. KENNEDY, San Francisco Mines of Mexico, Ltd., Apartado 48,
 Parral, Chih., Mexico.
 MACKAY, HENRY S. MacKay Copper Process Co., Norwich, Conn.
 MAGNUSON, MARK G. 49 Chamber of Commerce, Minneapolis, Minn.
 MARSHALL, JAMES J. R. F. D. 3, Farmville, Va.
 MARSHALL, S. M., Mgr., Turbine, Condenser and Pump Departments, Southwark
 Foundry & Machine Co., 5th & Washington Ave., Philadelphia, Pa.
 MASTERS, HARRIS K. 392 E. Washington Ave., Bridgeport, Conn.
 MAXON, WALTER LONDON. 60 Greenbush St., Cortland, N. Y.
 MEANS, ELLISON C. Box 99, Rockville, Md.
 MITINSKY, A. N., Min. Engr., Chief Engr. of Railroad supplies department,
 V. O. 2, Line N., 3, Petrograd, Russia.
 MOORE, EDWARD W. 5343 Dorchester Ave., Chicago, Ill.
 MOORE, ELWOOD S. Dept. of Geology, State College, Pa.
 MOORE, L. D. Care Darrow-Mann Co., 40 Central St., Boston, Mass.
 MOULE, J. W., Care Australian Institute of Mining Engineers,
 57 Swanston St., Melbourne, Vic., Australia.
 MUNROE, CHARLES E., Prof. of Chemistry, George Washington Univ.,
 Washington, D. C.
 NEWELL, ARTHUR. Darby, Mont.
 NEWELL, FREDERICK H. Head of Civil Engr. Dept., Univ. of Illinois, Urbana, Ill.
 NIEDING, BURTON B. Perseverance, via Thane, Alaska.
 NIGHMAN, C. EDWIN, Shift Boss, Leonard Mine, Anaconda Copper Min. Co.,
 Butte, Mont.
 ORMROD, JOHN D, Pres. Donaldson Iron Co., Emaus, Lehigh Co., Pa.
 PALMER, WILLIAM FREDERICK. Miami, Ariz.

- PARKER, EDWARD W., Director, Anthracite Bureau of Information,
1032 Miners Bank Bldg., Wilkes-Barre, Pa.
- PERKINS, H. C. 1701 Connecticut Ave., Washington, D. C.
- PETERSON, AXEL M. 810 Peach St., Erie, Pa.
- PHILLIPS, DRURY MCNEILL. 1220 Proctor St., Port Arthur, Tex.
- PHILLIPS, WILLIAM B., Pres. Colorado State School of Mines, Golden, Colo.
- PITMAN, S. M. Box 1118, Providence, R. I.
- POCOCK, CECIL W. Northern Ore Co., Edwards, N. Y.
- POTTER, WINFIELD S., Mfr., Pres. Alloy Steel Forgings Co., Carnegie, Pa.
- PROCHAZKA, GEORGE A., JR. Central Dye Stuff Chemical Co., Newark, N. J.
- READ, THOMAS T., Care Engels Copper Min. Co., Engels, via Keddie, Plumas Co., Cal.
- RHOADS, ALBERT E. Frontier Brass Co., Niagara Falls, N. Y.
- RIEBEL, OTTO F. 972 E. 27th St. N., Portland, Ore.
- ROBERTSON, JASPER T. 9541 Winchester Ave., Chicago, Ill.
- RODGERS, JOSEPH H. R. F. D. 10, Box 291, Los Angeles, Cal.
- RODGERS, SELDEN S. Care Union Basin Mining Co., Golconda, Ariz.
- ROGERS, BENJAMIN C. Detroit Copper Mining Co., Morenci, Ariz.
- RORR, FRANK CURTIS. Care Mine La Motte Co., Mine La Motte, Mo.
- ROSEWARNE, D. D., Royal Colonial Institute, Northumberland Ave., London,
W. C., England.
- RYAN, EDWARD C., Asst. Supt. Federal Furnace Co., S. Chicago, Ill.
- SCHABER, CARL F., Div. Engr., Tennessee Coal, Iron & R. R. Co., Bessemer, Ala.
- SCHAEFER, FREDERICK RULLMAN. Good Roads Machinery Co., Fort Wayne, Ind.
- SCHLEBE, M. C. Box 55, Sinton, Tex.
- SCHMALE, CHARLES H. 130 Burlington Ave., Billings, Mont.
- SEMPLE, ROBERT ALEXANDER. 57 W. 53d St., New York, N. Y.
- SENGER, RICHARD WARREN. Asarco, Durango, Mexico.
- SHEAFE, HARRY J. Instructed to hold all mail.
- SKOGMARK, JOHN, Constr. Engr. Mineral Paint Line Co., Chicago, Ill.
- SMALE, FRANK L. Box 289, Victor, Colo.
- SMITH, LLOYD B. 824 S. Detroit Ave., Tulsa, Okla.
- SOMERS, RANSOM E. 804 E. Seneca St., Ithaca, N. Y.
- SONNEMANN, G. A. 218 Riverside Ave., Spokane, Wash.
- SPARKS, JAMES T. 326 Erie St., El Paso, Tex.
- SPROAT, A. D., Cia. de Santa Gertrudis S. A., Pachuca, Hgo., Mexico, via Vera Cruz.
- STARKEY, TOM R. Penmaen, Hampton Wick, Middlesex, England.
- STAVER, W. H. Ophir, Colo.
- SWEENEY, HARRY P., Genl. Mgr., Canadian Atlantic Coal Co., Ltd.,
Port Malcolm, N. S., Canada.
- THOMPSON, ARTHUR P. Magdalena, N. M.
- THOMSON, ALEXANDER T., Care Phelps, Dodge & Co., 99 John St., New York, N. Y.
- THOMSON, EDWARD. Av. Vallarta 107, Guadalajara, Jal., Mexico.
- THORNE, WILLIAM E., Lenskoie Gold Mining Co., Ouspensky Mine, Bodaibo, Siberia.
- TIBBY, BENJAMIN F. 4 Caithness Apts., Salt Lake City, Utah.
- TSAI, HSIANG. Columbia University, New York, N. Y.
- WANG, CHUNG YU. Tayeh Iron Mines, Tayeh, Hupeh Province, China.
- WARNER, ROBERT K. 128 Oneida St., Syracuse, N. Y.
- WARREN, FRANK M., Director, Asst. Sec., Treas., Engr., Pine Land Co.,
1224 First "National-Soo" Line Bldg., Minneapolis, Minn.
- WEST, WILLIAM C. Fort Myers, Fla.
- WICKS, FRANK R., Asst. Supt. Chino Copper Co., Hurley, N. M.
- WILDING, JAMES, Met. and Min Engr., Mineral Separation American
Syndicate, 825 Merchants Exchange Bldg., San Francisco, Cal.
- WILSON, GORDON. Apartado 7, Zacatecas, Zac., Mexico.
- WITT, HERBERT NELSON. Care A. M. Ardery, Carson City, Nev.

ADDRESSES OF MEMBERS AND ASSOCIATES WANTED

- | Name | Last address of Record, from which Mail has been Returned. |
|------------------------|--|
| BOISE, CHARLES WATSON, | Care Forminière, 8 Montagne du Parc, Brussels, Belgium. |
| CLERC, CAMILLE. | 92 Rue Jouffrey, Paris, France. |
| DOWLER, HARRY P. | Penn-Mary Coal Co., Heilwood, Pa. |
| GEIGER, ARTHUR W. | 220 Mills Bldg., San Francisco, Cal. |
| GRAVES, McDOWELL. | 949 Atlantic Ave., Long Beach, Cal. |
| JONES, THOMAS J. | Perm Govt., via Petrograd, Russia. |

KIRKLAND, T. C., Cia. Met. y Refinadora del Pacifico S. A., Fundicion, Son., Mexico.	
McCARRICK, E. 628 N. Serrano Ave., Los Angeles, Cal.	
MILLER, FRANK BARTON. Blair, Esmeralda Co., Nev.	
MORRIS, CHARLES E. Pony, Madison Co., Mont.	
MORSE, ROBERT G. Mass Steel Casting Co., Everett, Mass.	
OLIVER, ROLAND B. 8 Montagne du Parc, Brussels, Belgium.	
PRICHARD, WILL A. Room 711, 111 Broadway, New York, N. Y.	
RHODES, W. B. Golden, Colo.	
RUBEL, MILTON L., Pilonas Min. Co., Topia, Dgo., Mex., via Nogales	
	and Culiacan, E. de Sin.
SCHINDLER, DONALD F. 215 N. Murray St., Madison, Wis.	
SEIBERT, PERCY A. Hagerstown, Md.	
SOMERS, RANSOM E. 804 E. Seneca St., Ithaca, N. Y.	
STILLWAGON, SAM C. 329 W. 58th St., New York, N. Y.	
WELSH, NORVAL J. E. Organ, P. O., N. M.	
WILCOX, RALPH. Curanilahue, Chile.	
WRIGHT, JESSE T. Wallace, Idaho.	

NECROLOGY

The deaths of the following members were reported to the Secretary's office during the period July 10 to Aug. 10, 1915:

Date of Election.	Name.	Date of Decease.
1886	* Foote, William Banks.	June 25, 1915
1887	* Green, Charles B.	Apr. 24, 1915
1895	* Holmes, Joseph A.	July 12, 1915
1881	* Ormrod, George.	June 21, 1915

* Member

EXECUTIVE COMMITTEES OF LOCAL SECTIONS

New York

Meets first Wednesday after first Tuesday of each month.

DAVID H. BROWNE, *Chairman*, JOHN H. JANEWAY, *Vice-Chairman*.
 F. E. PIERCE, *Secretary*, 35 Nassau St., New York, N. Y.
 P. A. MOSMAN, *Treasurer*.

LEWIS W. FRANCIS.

Boston

Meets first Monday of each winter month.

HENRY L. SMYTH, *Chairman*, ALFRED C. LANE, *Vice-Chairman*.
 HENRY A. WENTWORTH, *Secretary-Treasurer*, 60 India St., Boston, Mass.
 ROBERT H. RICHARDS, ALBERT SAUVEUR.

Columbia

Holds four sessions during year. Annual meeting in September or October.

FRANK A. ROSS, *Chairman*, RUSH J. WHITE, *Vice-Chairman*.
 LYNDON K. ARMSTRONG, *Secretary-Treasurer*, P. O. Drawer 2154, Spokane, Wash.
 FREDERIC KEFFER, FRANCIS A. THOMSON.

Puget Sound

Meets second Saturday of each month.

I. F. LAUCKS, *Chairman*, J. F. MENZIES, *Vice-Chairman*.
 GLENVILLE A. COLLINS, *Secretary-Treasurer*, Box 144, Seattle, Wash.
 H. L. MANLEY.

Southern California

SEELEY W. MUDD, *Chairman*, C. COLCOCK JONES, *Vice-Chairman*.
 FREDERICK J. H. MERRILL, *Secretary-Treasurer*, 631 Higgins Bldg., Los Angeles, Cal.
 RALPH ARNOLD, A. B. CARPENTER.

Colorado

CHARLES A. CHASE, *Chairman*, S. A. IONIDES, *Vice-Chairman*.
 C. LORIMER COLBURN, *Secretary-Treasurer*, 614 Ideal Bldg., Denver, Colo.
 FRED H. BOSTWICK, W. G. SWART.

Montana

FRANK M. SMITH, *Chairman*, JAMES L. BRUCE, *Vice-Chairman*.
 DARSIE C. BARD, *Secretary*, Montana State School of Mines, Butte, Mont.
 FREDERICK LAIST, W. C. SIDERFIN.

San Francisco

Meets second Tuesday of each month.

G. HOWELL CLEVINGER, *Chairman*, C. W. MERRILL, *Vice-Chairman*.
 JAMES C. RAY, *Secretary-Treasurer*, 1235 Webster St., Palo Alto, Cal.
 F. W. BRADLEY, ANDREW C. LAWSON.

*Pennsylvania Anthracite Section*R. V. NORRIS, *Chairman*.

CHARLES F. HUBER, *Vice-Chairman*, W. J. RICHARDS, *Vice-Chairman*,
 EDWIN LUDLOW, *Vice-Chairman*, ARTHUR H. STORRS, *Vice-Chairman*.
 CHARLES ENZIAN, *Secretary*, U. S. Bureau of Mines, Wilkes-Barre, Pa.
 DOUGLAS BUNTING, FRANK A. HILL, ALBERT B. JESSUP.
 RUFUS J. FOSTER, JOHN M. HUMPHREY, ROBERT A. QUIN.

St. Louis

ARTHUR THACHER, *Chairman*, R. A. BULL, *Vice-Chairman*.
 WALTER E. McCOURT, *Secretary-Treasurer*, Washington Univ., St. Louis, Mo.
 H. A. BUEHLER, R. R. S. PARSONS, HERBERT A. WHEELER.

Chicago

ROBERT W. HUNT, *Chairman*, J. A. EDE, *Vice-Chairman*.
 HENRY W. NICHOLS, *Secretary-Treasurer*, Field Museum of Natural History, Chicago, Ill.
 F. K. COPELAND, G. M. DAVIDSON.

Utah

R. C. GEMMELL, *Chairman*, C. W. WHITLEY, *Vice-Chairman*.
 ERNEST GAYFORD, *Secretary-Treasurer*, 159 Pierpont Ave., Salt Lake City, Utah.
 WALTER FITCH, WILLIAM WRAITH.

STANDING COMMITTEES

*Executive*WILLIAM L. SAUNDERS, *Chairman*GEORGE D. BARRON,
SIDNEY J. JENNINGS,JOSEPH W. RICHARDS,
BENJAMIN B. THAYER.*Membership*JOHN H. JANEWAY, *Chairman*KARL EILERS,
LEWIS W. FRANCIS,LOUIS D. HUNTOON,
ARTHUR L. WALKER.*Finance*GEORGE D. BARRON, *Chairman*.

ALBERT R. LEDOUX,

CHARLES F. RAND.

*Library*E. GYBBON SPILSBURY, *Chairman*.⁴KARL EILERS³ALEX C. HUMPHREYS,⁴JOHN HAYS HAMMOND,¹

BRADLEY STOUGHTON.

*Papers and Publications*BRADLEY STOUGHTON, *Chairman*.

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WILLIAM H. BLAUVELT

HENRY M. HOWE,

ROBERT H. RICHARDS,

H. A. BRASSERT,

LOUIS D. HUNTOON,

L. D. RICKETTS,

WILLIAM CAMPBELL,

J. E. JOHNSON, JR.,

HEINRICH RIES,

R. M. CATLIN,

LEE O. KELLOGG,

E. F. ROEBER,

ALLAN J. CLARK,

WILLIAM KELLY,

RENO H. SALES,

FREDERICK G. COTTRELL,

JAMES F. KEMP,

ALBERT SAUVEUR,

NATHANIEL H. EMMONS,

CHARLES K. LEITH,

HENRY L. SMYTH,

JOHN W. FINCH,

ANTHONY F. LUCAS,

A. A. STEVENSON

CHARLES H. FULTON

EDWARD P. MATHEWSON,

RALPH H. SWEETSER,

F. LYNWOOD GARRISON,

HERBERT A. MEGRAW,

FELIX A. VOGEL,

ROBERT C. GEMMELL,

RICHARD MOLDENKE,

ARTHUR L. WALKER,

CHARLES W. GOODALE,

SEELEY W. MUDD,

ROLLA B. WATSON,

HARRY A. GUESS,

R. V. NORRIS,

HORACE V. WINCHELL,

R. DAWSON HALL,

EDWARD W. PARKER,

FREDERICK W. WOOD,

PHILIP W. HENRY,

EDWARD D. PETERS,

DWIGHT E. WOODBRIDGE.

HEINRICH O. HOFMAN,

ROSSITER W. RAYMOND,

COMMITTEE ON JUNIOR MEMBERS AND AFFILIATED STUDENT SOCIETIES

HARRY H. STOEK, *Chairman*.*Vice-Chairmen*

CHARLES H. FULTON,

FREDERICK W. SPERR,

EDWIN C. HOLDEN,

GEORGE J. YOUNG,

WALTER R. CRANE, *Secretary*, Pennsylvania State

College, State College, Pa.

LUTHER W. BAHNEY,

HOWARD ECKFELDT,

CHARLES J. NORWOOD,

DARSIE C. BARD,

DAVID M. FOLSOM,

GEORGE S. RAYMER,

ROBERT H. BRADFORD,

R. R. GOODRICH,

HEINRICH RIES,

SAMUEL W. BEYER,

CHARLES E. LOCKE,

HENRY L. SMYTH,

GUY H. COX,

JAMES F. MCCLELLAND,

FRANCIS A. THOMSON,

JOSEPH DANIELS,

HARRY B. MELLER,

F. W. TRAPHAGEN,

NOAH F. DRAKE,

WALTER S. MORLEY,

CLINTON M. YOUNG.

FRANK W. DURKEE,

HENRY S. MUNROE,

¹ Until Feb., 1916.² Until Feb., 1917.³ Until Feb., 1918.⁴ Until Feb., 1919.

COMMITTEE ON INCREASE OF MEMBERSHIP

THOMAS T. READ, *Chairman.*DAVID H. BROWNE, *First Vice-Chairman.**Vice-Chairmen*

LYNDON K. ARMSTRONG,
EDWARD H. BENJAMIN,
CHARLES A. BUCK,
H. W. HARDINGE,
DUNCAN MACVICHIE,

VAN H. MANNING,
HENRY W. NICHOLS,
ERSKINE RAMSAY,
SUMNER S. SMITH,
WALTER G. SWART.

EDWARD L. DUFOURCQ, *Secretary*, C 22 Produce Exchange, New York, N. Y.

HUNTINGTON ADAMS,
F. L. ANTISELL,
FRANKLIN BACHE,
EDWIN G. BANKS,
PERCY G. BECKETT,
LOUIS V. BENDER,
JAMES G. BERRYHILL,
ALBERT C. BOYLE, JR.
D. H. BRADLEY, JR.
VICTOR M. BRASCHI,
FREDERIC K. BRUNTON,
LESTER R. BUDROW,
MILTON A. CAINE,
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CHARLES CATLETT,
NOAH F. DRAKE,
J. A. EDE,
LOUIS V. EMANUEL,
AUGUSTUS H. EUSTIS,
WALTER FITCH,
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STEPHEN L. GOODALE,
CARL E. GRUNSKY, JR.,

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RICHARD S. HASELTINE,
PHILIP W. HENRY,
FRANK R. HEWITT,
JOSEPH T. HILLES,
JOHN HOATSON,
ROY J. HOLDEN,
TADASHIRO INOUE,
YING-CHIEH KUANG,
CLANCY M. LEWIS,
I. P. LIHME,
JOHN J. LINCOLN,
DOUGLAS C. LIVINGSTON,
SPENCER R. LOGAN,
BRUNO V. NORDBERG,
HENRY M. PARKS,
RICHARD C. PATTERSON, JR.
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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Important Topping Plants of California

BY ARTHUR F. L. BELL,* SAN FRANCISCO, CAL.

(San Francisco Meeting, September, 1915)

PRIOR to 1908 the oil production in the State of California had been almost entirely a heavy fuel oil, with a high flash point, but changed within a short period to a large percentage of refining oil with a low flash, by reason of the coming in of the Santa Maria field, the Midway field, and the increased production of the Fullerton field.

This is clearly shown in the following tabulation of California's production:

Year	Barrels	Below 20° Bé, Per Cent.	Above 20° Bé, Per Cent.
1908.....	48,307,000	90	10
1910.....	77,698,000	80	20
1912.....	90,075,000	69	31
1914.....	103,600,000	47	53

The bringing in of these high percentages of light oil with their dangerous low flash and the increasing demand for the refined products necessitated the devising of some cheap and efficient means of quickly separating the lighter elements either in the field or at the main distributing points; also it became necessary to remove the excessive percentage of water which many of the light oils carried in emulsion.

UNION OIL CO.

Port Harford Plant

Probably the first important attempt to top light oil in California by other means than the ordinary stills was tried about 1908 by the Union Oil Co., at its plant at Port Harford, San Luis Obispo County, with Santa Maria oils. The process was the invention of Hubert G. Burroughs who afterwards obtained a patent for the process No. 998837, dated July

*Chief Engineer, Associated Oil Company.

25, 1911. This consisted of a series of 12-in. pipes mounted in units of three pipes, one above the other, the crude oil being admitted to the upper pipe and filling the pipe about two-thirds of its diameter at which point it overflowed to the next 12-in. pipe below and in the same manner the oil again flowed to the third pipe below. Super-heated steam was the heating agent, which passed through endless coils in the lower half of the 12-in pipes. The oil was heated to about 200° F. in the upper pipe and reached a final heat of about 325° F. in the bottom pipe. The vapors generated passed off through the vapor line to the condenser.

The gravity of the Santa Maria crude, when the plant started, ran from 23 to 24 B $\acute{e}$, and averaged only from 2 to 3 per cent. water, but since then the average percentage of water has increased to 25 to 30 per cent.

This plant was operated for two or three years but the system developed several objectionable features. It was not economical, and by reason of the increase in water contents, the capacity was limited and was therefore abandoned.

Avila Plant

A new plant was designed and erected under the supervision of E. I. Dyer, Engineer in Chief of the Union Oil Co. This plant, of which A. Roy Heise is superintendent, is at Avila, near Port Harford, San Luis Obispo County, about 40 miles from the Santa Maria field, the oil being delivered to the plant from the field by two pipe lines; one 6 in. and the other 8 in. This plant, although erected several years ago, is still one of the best designed topping plants on the Coast. It is topping oils in some cases carrying as high as 20 per cent. water at the same cost for fuel and labor as some other plants topping oils carrying only 2 or 3 per cent. water.

The plant has a capacity of about 10,000 bbl. per day of clean oil or, as the oil is now running, 8,000 bbl. of net oil carrying about 20 per cent. water. The oil coming from the field carries as high as 30 per cent. of water, but a portion of it is directly treated in an electric apparatus, manufactured by the Petroleum Rectifying Co. of California, the oil thus treated being brought down to about 5 per cent., which when mixed with the rest of the oil coming from the field, gives an average of about 20 per cent. of water in the oil handled by the topping plant.

The heat used in topping is steam of 150 lb. pressure generated by a battery of five 300-h.p. Stirling boilers in a separate boiler house.

There are 12 cylindrical stills mounted in a steel-frame construction in a separate building, the stills being mounted in a double row of six each, one row above the other. Those in the upper set are known as the low-pressure stills, those in the lower as the high-pressure stills. Live steam is admitted to the lower row only. (Fig. 1.)

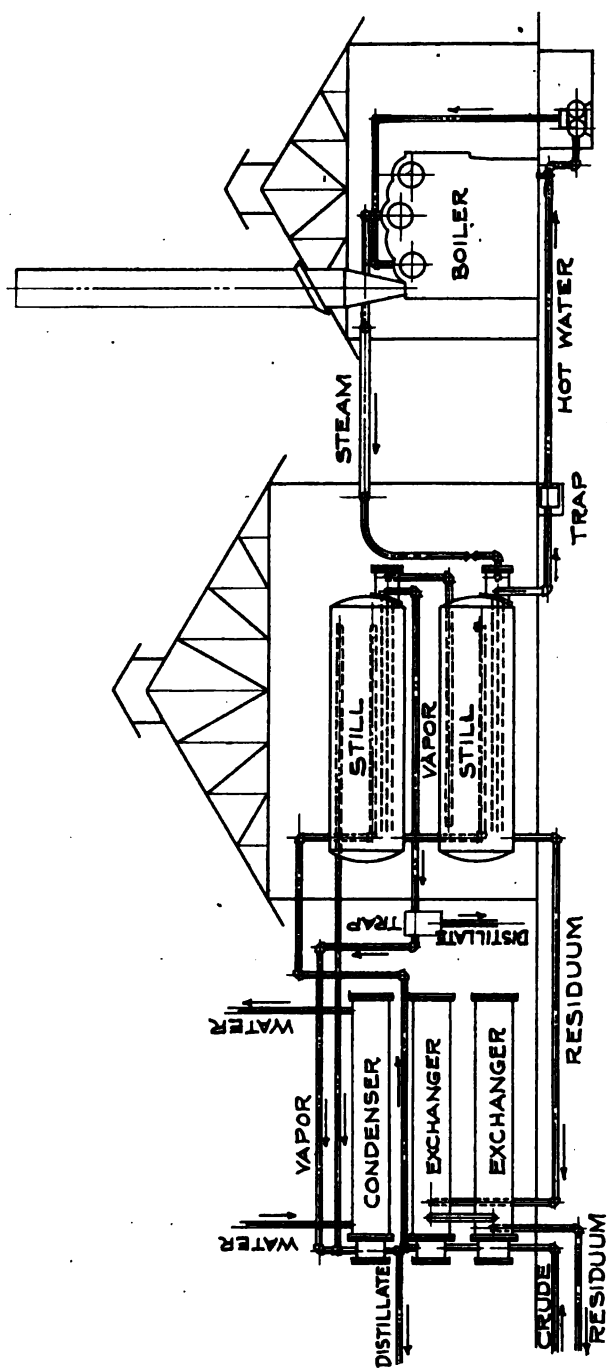


FIG. 1.—UNION OIL CO. PLANT, AVILA, SAN LUIS OBISPO COUNTY, CALIFORNIA.

Each still is about 6 ft. in diameter and 20 ft. long and has mounted on its front end, a steam chest, having three separate vertical compartments, the live steam being admitted to one compartment and passing to the next compartment through a series of 1-in. continuous pipes which travel from the initial side of the steam chest, the full length of the still and return to the next compartment of the steam chest, and in the same manner to the third compartment, so that there is no connection of live steam between any of the compartments of the steam chest except through the heating coils. The condensed water is trapped out of each compartment so that the minimum quantity of condensed water travels through the coils.



FIG. 2.—GENERAL VIEW OF THE AVILA PLANT.

As the steam is condensed, it drains to traps and is returned to the boilers at 280° F. No live steam is admitted to the upper tier, or low-pressure stills, the heating agent being the evaporated vapors from the bottom still which rise and enter the steam chest of the upper still and pass through in the same manner as the live steam entered the lower still. The vapor pressure in both the lower still and inside the coils of the upper still ranges from 40 to 50 lb., the pressure being maintained at a point where the radiation of the latent heat through the coils to the liquid in the upper still is equalized by the heat vapors rising from the lower still. As about half of the vapor is steam, the temperature corresponds closely to that of saturated steam of the same pressure. No vapor leaves the coils of the upper still; nothing but the condensed liquids are trapped off, which go to the same condenser as the vapors from the low-pressure still. By this means every degree of latent heat in both the oil and water vapors rising from

the high-pressure still is absorbed by the oil and water evaporated from the low-pressure still, thereby effecting a saving of about 50 per cent. of the fuel necessary to top the oil in ordinary stills. The vapors from each low-pressure still flow into the condenser belonging to that battery, which is mounted on the top of a steel construction outside of the steam-still house.

There are six separate condensers, each connecting to a battery of a high- and a low-pressure still. The general design of the condensers is rectangular with a vapor chest at the rear end of the same general design as that of the stills, except that the dividing partition in the chest is horizontal. The vapors are admitted to the upper portion of the chest and pass through a series of 2-in. return pipes, surrounded by the condensing water, and return to the lower portion of the chest, from there draining to the proper storage tanks.

Mounted below each condenser are two heat exchangers set one above the other. These heat exchangers are similar in construction to the condenser except that they have 1-in. return pipes, and have two vertical dividing partitions in each chest; also heated residuum surrounds the pipes instead of water. The heated residuum from the high-pressure still passes into one end of the body of the upper exchanger and travels backward and forward four times, guided by interior partitions. It then passes into the end of the body of the lower exchanger, again traveling backward and forward four times before passing out as finished residuum to storage. In all condensers and exchangers, the liquids counterflow.

The crude oil in taking up the heat in the exchangers comes into one side of the chest of the lower exchanger, passes backward and forward twice through a series of 1-in. return pipes and then is discharged from the opposite side of the chest to the upper exchanger. It passes through two series of return pipes in the upper exchanger, leaving the chest on the opposite side from where it entered. It flows to the low-pressure still, and after giving off a proportion of its vapors of oil and water, is pumped to the high-pressure still where all of the remaining water and distillates are taken off. The residuum oil passes to the exchangers.

When I examined these stills, the pressure of the vapors rising from the low-pressure stills was 3.6 lb. with a temperature of 183°F. In the high-pressure stills, and inside the coils of the low-pressure stills, the vapor pressure was 48.3 lb. with a temperature of 279° F.

This is a most economical system, since the only steam admitted to the plant is through the high-pressure stills; all of the work done in the low-pressure stills is therefore an absolute saving.

The plant is running on what is known as double effect, but at times has been run on triple effect, that is, live steam was introduced into the first still only and the heated vapor from the first still under pressure was admitted to the coils of the second still, and the heated vapor from the

second still, necessarily under a less pressure, was admitted to the coils of the third still, the crude oil flowing first to the low-pressure, then to the intermediate, and finally to the high-pressure still.

Distillation by Double Effect

	Water, Per Cent.	Distillate, Per Cent.
Evaporation from low-pressure still.....	56.95	34.45
Evaporation from high-pressure still.....	42.05	64.15
Evaporation, Interchanger and Unaccounted.	1.00	1.40
	100.00	100.00

This method, as would be expected, showed a greater saving than the double effect, but reduced the quantity of crude that could be put through in a given time:

Distillation by Triple Effect

	Water, Per Cent.	Distillate, Per Cent.
Evaporation from low-pressure still.....	55.80	37.42
Evaporation from intermediate still.....	29.75	31.36
Evaporation from high-pressure still.....	13.42	29.74
Evaporation from interchanger and unaccounted.	1.03	1.48
	100.00	100.00

From the above figures, it will be seen that with the double-effect arrangement about 50 per cent. of the fuel is saved, and for triple effect about 75 per cent., over a single system of evaporation.

For a given period, the average temperature of the crude entering the heat exchangers was 61.67° F.; leaving the heat exchangers before entering the low-pressure stills, the temperature was 141.45° F., showing a saving by rise in temperature due to exchangers of 79.78° F.

The economy of the plant is shown by the following:

Duty Records per Barrel of Dry Fuel Oil Used for the Last Three Months of 1914

Barrels of crude oil treated.....	63.35
Barrels of water actually removed from crude.....	10.67
Barrels of residuum produced.....	42.34
Barrels of distillate produced.....	8.54
Barrels of residuum pumped from plant.....	43.80
Barrels of distillate pumped from plant.....	9.52
Barrels of circulating water pumped.....	156.46
Barrels of miscellaneous oil pumped.....	9.08
Kilowatt hours—lighting, power, etc.....	2.53

Thermal Duty of 1 bbl. of Dry Fuel Oil

	B.t.u.
Evaporating water from crude.....	4,135,984
Evaporating distillate from crude.....	425,128
Kilowatt hours—lighting and power.....	35,940
Total useful duty.....	4,597,052

The average percentage of water in the crude was 18.71 per cent.

A record test run on "Double Effect" showed the following results:

Fuel consumed per net barrel of:	Per Cent.
Crude treated.....	1.75
Residuum manufactured.....	2.51
Distillate manufactured.....	11.70
Water evaporated.....	11.58
Both water and dist. evaporated.....	5.82
Water in the crude oil.....	15.38

This system is ideal for Santa Maria oils and entirely eliminates the coking which takes place when direct heating is used under ordinary stills or retorts. The water carried by the oil is as salty as sea water and when evaporated throws down salt crystals in the stills. In the case of fire-heated stills, this salt mingles with the coke and forms a hard incrustation, which necessitates closing down frequently and chipping out the coke; whereas with this form of still, when salt crystals are formed, the operators employ the unique method of shutting down the stills about once a week, emptying the stills of oil and filling with hot water, thus dissolving the salt and passing it off by pumping a continuous stream of heated water through the stills. Two patents on this system have been applied for by Mr. Dyer and are now pending in the patent office.

Brea Refinery

The Union Oil Co. has two other topping plants located in the Fullerton field. The most important is the Brea refinery. The first portion of this plant was completed in August, 1911, to handle the light oils from the Birch well, which came in with a very gassy oil flowing about 2,400 bbl. per day. It consisted of eight old 40-h.p. drilling boilers, with half the tubes removed; these boilers were set in two rows of four facing each other with an alley between, the oil being run through each battery of four boilers continuously. The plant when first started treated successfully about 100,000 bbl. per month of clean dry oil carrying not more than 0.5 per cent. of water, removing 22 per cent. of 53° tops. However, it has since developed that this plant is not adapted to handling wet oils and will

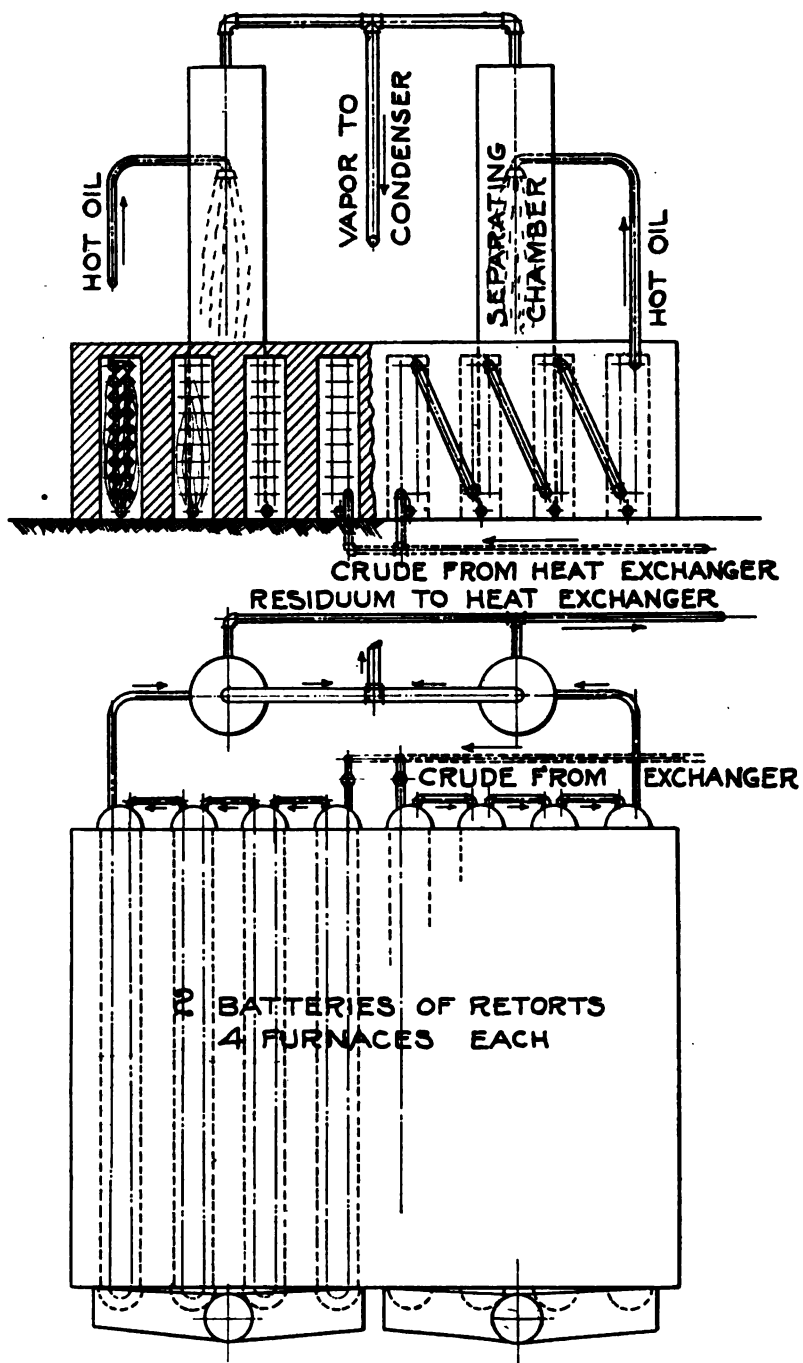


FIG. 3.—TOPPING PLANT AT BREA REFINERY, UNION OIL CO.

not work economically with oils carrying more than 2 per cent. water. It is therefore kept running on the cleaner oils.



FIG. 4.—FRONT VIEW OF BREA TOPPING PLANT.

The next step in increasing the capacity of the plant was the building of an additional topping plant in March, 1913. This consisted of two



FIG. 5.—REAR VIEW, SEPARATORS IN BACKGROUND.

batteries of retorts, each battery being divided into two separate divisions, each division having four furnaces and each furnace carrying a double row,

9 pipes high, of 3-in. return pipes, 20 ft. long. Each division of the battery is independently fed with crude oil. The oil admitted to the bottom pipe of the furnace travels up through the nine coils, coming out the top and returning to the bottom pipe of the next furnace, again leaving the upper pipe and so on through the four furnaces. The heated oil from each side of the battery flows by independent pipes to one of the separators. This makes each 8-furnace battery and one separator a single unit.

The vapors are then led from the separating chambers to the condensers and on to the storage tanks.

The residuum is taken from the bottom of the separators, passed through the heat exchangers and on to the residuum storage.



FIG. 6.—REARRANGED FURNACE FOR OIL BURNING.

The coils in the furnace are heated by gas fed through perforated pipes running the full length of the furnace. It was found that this form of furnace is not well adapted to oil firing. A change was, therefore, made in one of the batteries, the coils being mounted horizontally in a brick furnace, allowing plenty of combustion room below. This furnace is working very satisfactorily and it is likely that the other half of the battery will have the coils changed from vertical to horizontal position so as to be adapted to oil firing. (Fig. 6.)

The total capacity of the two batteries of retorts with gas fires was 60,000 bbl. per month of 21 to 22° Bé. oil, carrying 6 per cent. B. S. & M., from which was topped about 10 per cent. of 51 gravity distillate, giving a residuum of 18° Bé. and from 160 to 180 flash. With the change in construction and the use of oil firing, it is believed the capacity will be doubled.

The hot oil at the outlet end of the retort is choked down to a pressure of 40 lb., but flows into the separator at practically atmospheric pressure. This choking of the oil gives a back pressure at the inlet end, due to friction in the retort, of 80 lb.

The crude oil leaves the heat exchanger and enters the first coil of the retort at 180° F.; it leaves the coils and enters the separators at about 450° F.

This refinery not only tops oil but turns out finished gasoline, but as the purpose of this paper is to describe topping plants only, no description will be given of the other details of the plant.

Naranjal Topping Plant

The Naranjal reduction or topping plant of the Union Oil Co., is in section 7, T. 3 S., R. 9 W., in the Fullerton field. This is the smallest of the three Union topping plants, having a maximum capacity of 2,200 bbl. per day. The oils treated are 16° to 18° Bé. and tops recovered 56° Bé. The plant has been built about 4 years and is known as a Brown-Pickering topping plant, it having been designed by A. H. Brown, the Superintendent. (Figs. 7, 8, 9.)

The plant consists of two independent units, each unit having a rectangular still about 4 by 4 by 20 ft. with cover bolted on and bottom of still fitted with cross baffles. The still is set, with a fall of about 6 in., over a brick furnace in which are placed 13 3-in. retort pipes, having return bends so as to make a continuous coil. The hot gases from the still furnace pass through an underground flue to a 40-h.p. drilling boiler, bricked up in the ordinary manner, these gases passing first under the boiler then through the tubes and out of the stack. In addition, the ordinary field type of a heat exchanger is used, made of a small pipe inside of a large one; for cooling the vapors, a series of wooden condenser boxes with pipe coils inside are provided.

In operating the plant, the crude oil is pumped through the inner pipe of the heat exchanger, the hot residuum flowing between the two pipes. The partially heated crude oil passes through the boiler, heated by waste gases, and from there through the coil retort in the furnace under the rectangular still, then into the high end of the still, where it is released and allowed to flow back and forth guided by the baffles until it reaches

the lower end of the still, from which it flows by pipe through the heat exchanger to the residuum storage. As the oil flows through the still,

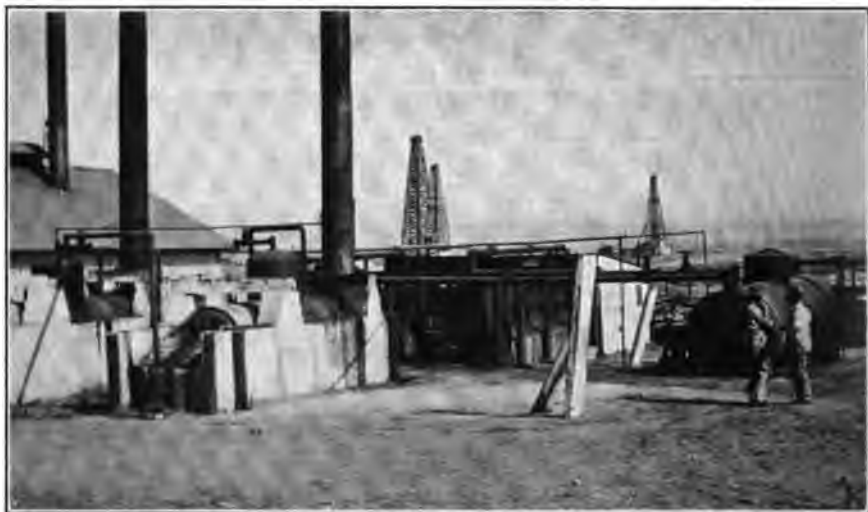


FIG. 7.—SIDE VIEW OF THE NARANJAL PLANT, SHOWING BOILERS AND STILLs.



FIG. 8.—STILLS.

which is heated by fire on its under side, the vapors pass off to a separating chamber, where the heavier vapors condense and are drawn off, and the lighter vapors pass to the condensers.

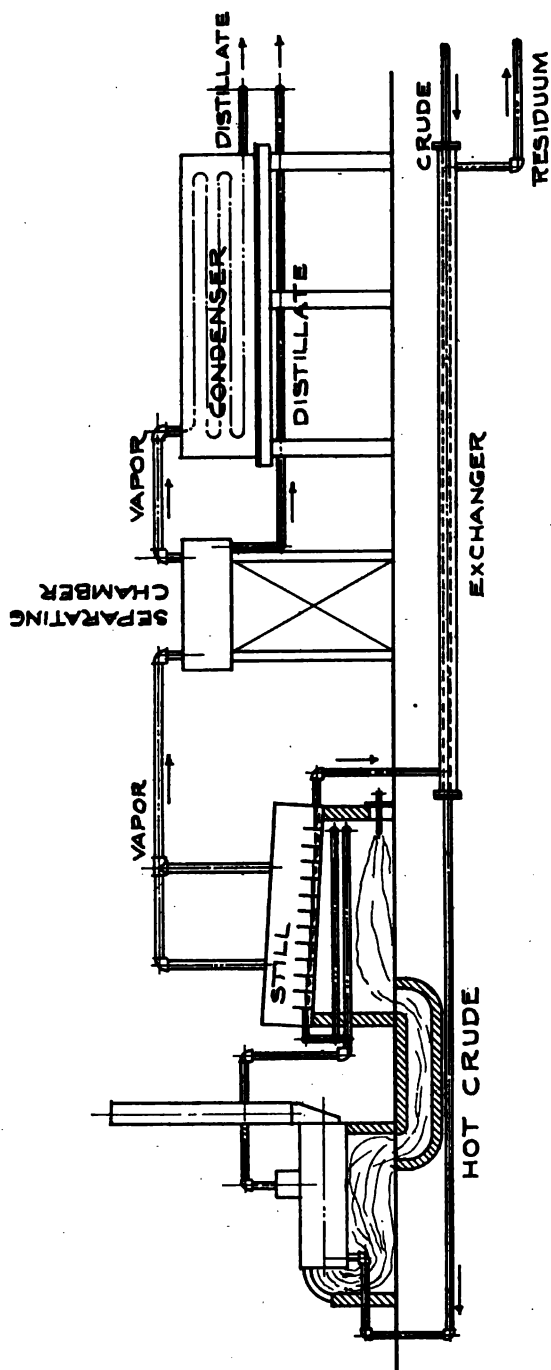


FIG. 9.—PLANT OF THE UNION OIL Co., NARANJAL, FULLERTON FIELD. BROWN-PICKERING DESIGN.

Each unit is a duplicate of the other, but is run entirely separate up to where the vapors enter the separating chamber; from there on the mixed vapors pass to the condensers. This system carries the notice "Patent applied for."

A number of these Brown-Pickering plants have been constructed in the oil fields and have given fair results with clean oil. They are not well adapted for wet oil, for as soon as they are worked to capacity with wet oil foam appears, which discolors the distillate, and the construction of the still does not admit of putting any pressure on it to hold down the foam.

STANDARD OIL COMPANY

On the Murphy, Coyote Hills, property, Fullerton field, since bought by the Standard Oil Co., is one of the Brown-Pickering topping plants, consisting of four rectangular stills about 4 by 4 by 10 ft., each set with a gradual fall so that the oil admitted into the upper end of the first still flows down to its lower end and out into the upper end of the next still, which is set beside it and so on until the oil had flowed by gravity through the four stills.

At the rear end of the four stills is erected one 40-h.p. boiler so constructed that the hot gases from the four still furnaces pass under the boiler and through the tubes and out of the boiler stack. The vapors from the first two stills were run together and kept separate from the vapors coming off the third and fourth stills which were also run together. The vapors were then each run through separate pipe heat exchangers, the vapors in the inside pipe and the crude between the inner and outer pipes.

The continuation of the inside vapor pipe was then returned toward the plant and surrounded by an outer pipe this time filled with water, this forming the condenser for the vapors. The condensed vapors were then run to storage tanks.

In operating, the oil was first pumped through the outside of the vapor heat exchanger, then through a residuum heat exchanger, then to the boiler and from there to the upper end of the first still, then out of the lower end of the last still to residuum storage.

This plant had a capacity of about 1,200 bbl. per day of clean oil and 600 to 650 bbl. of wet oil carrying 25 per cent. water. If this latter capacity was exceeded, foaming took place.

With the crude oil entering the vapor heat exchanger at about 60° F., it entered the boiler heater at 80°, and left the boiler to enter the first still at 90° F.

The temperature of the oil on leaving the four stills ran 175° F., 200° F., 220° F., and 240° F., respectively. The fuel consumed was gas.

After the Standard Oil Co. purchased the property, the plant was used

as a dehydrator, but was found so expensive that a 6-treater electric plant of the Petroleum Rectifying Co. was installed, which cleans the oil more satisfactorily and at a much less cost.

RICE RANCH OIL COMPANY

Another Brown-Pickering plant of the same type, but of a single unit, has been erected by the Rice Ranch Oil Co. in the Santa Maria field, Santa Barbara County. This plant is used mostly as a dehydrating plant but has also been used for topping. It was erected about 1911 and has worked very satisfactorily. Its capacity is claimed to be about 650 bbl. of crude carrying 3 per cent. water, but it also handles, at a reduced capacity, oil carrying as high as 30 per cent. water. The course of the oil through the plant is the same as in the other Brown-Pickering plants.

AMERICAN OIL FIELDS COMPANY OF CALIFORNIA

Another important topping plant is that erected by the American Oilfields Co., of California, about 1910, on section 36-31-22, in the Midway

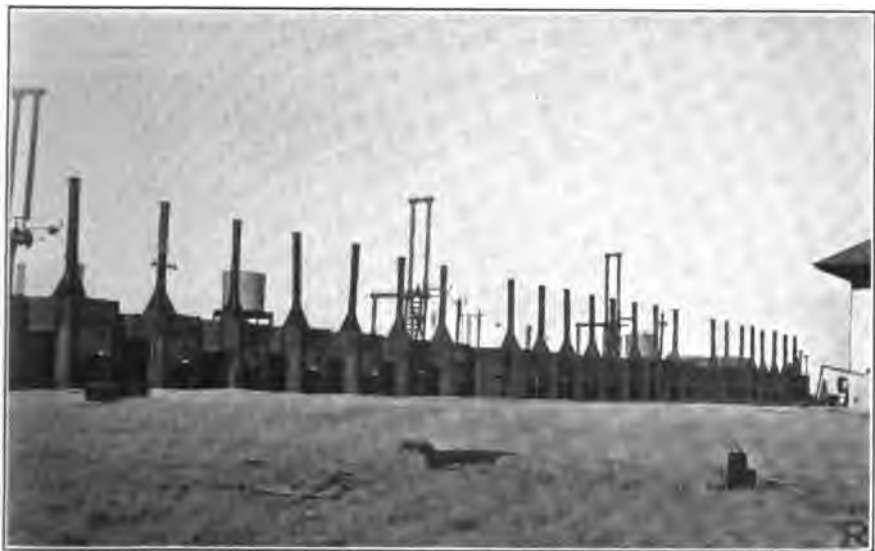


FIG. 10.—PLANT OF AMERICAN OIL FIELDS CO., SHOWING FRONT OF RETORTS.

field. This plant has an economical capacity of from 18,000 to 20,000 bbl. of crude per day. It is arranged with three batteries, each battery consisting of eight independent retorts, making 24 retorts used on crude oil. Each battery delivers its heated oil to a single separator, where the

vapors are separated, the vapors being led to a common header which distributes them to a series of independent condensers. (Figs. 10, 11, 12.)



FIG. 11.—REAR VIEW OF AMERICAN OIL FIELDS PLANT, WATER COOLING TOWERS IN BACKGROUND.



FIG. 12.—CONDENSERS OF AMERICAN OIL FIELDS PLANT.

This plant is well constructed and economically run and the company is to be complimented for its neat appearance.

All the condensing water is saved and run back through four independent cooling towers, which have a remarkable efficiency, since the circulating water after passing through these towers is cooled far below the temperature of the atmosphere. When I took observations, the water entered the towers at 140° F. and was taken out of the bottom at 65° F. with the surrounding temperature 99° F.

The plant is not only well adapted to topping clean crude, but is often called on to handle oils carrying as high as 25 and 30 per cent. water.

The fuel used at the refinery is natural gas.

PLANT OF SANTA FE RAILROAD CO., FELLOWS

The first really successful Trumbull plant, and the one that first attracted attention was erected at Fellows, Kern County, for the Santa Fe R. R. Co., to top its light oils so as to make safe fuel for locomotives. This plant consisted of two units of six 12½ in. by 20 ft. retorts, both units discharging into the one vapor separator.

The records of this plant for January, 1912, showed for 31 days' operation, with one of the units off for 5 days:

	Barrels	
Crude delivered to the plant.....	120,760.67	21° Bé.
Tops taken off.....	14,772	12.25 per cent.
Gasoline recovered.....	36.60	0.03 per cent.
Fuel consumed.....	1,575	1.34 per cent.

The Santa Fe R. R. Co. has since contracted its light oils to the General Petroleum Co. and has, therefore, closed down this plant.

PACIFIC CRUDE OIL CO., MIDWAY FIELD

The Pacific Crude Oil Co. built a small Trumbull plant on its property near Fellows for the purpose of topping oil in the field. Subsequently the company contracted its oil, so the plant was never started up. A good example of a small Trumbull topping plant is illustrated in Figs. 13 and 14.

GENERAL PETROLEUM CO., VERNON

The General Petroleum Co., in 1912, completed an 8-in. line from the Midway oil fields, Kern County, to San Pedro, Los Angeles County, and as the oils transported were too light and valuable for fuel purposes, it was necessary that a topping plant be erected at or near the terminal. The company adopted the Trumbull system of topping and erected a plant of 15,000 bbl. daily capacity (Figs. 15 to 19A) at the town of Vernon, in the outskirts of Los Angeles. All incoming oil is topped here and the residuum pumped to San Pedro. However, while this plant has a rated capacity of 15,000 bbl. daily, it has since been determined that the most economical capacity of the plant is about 12,000 bbl.

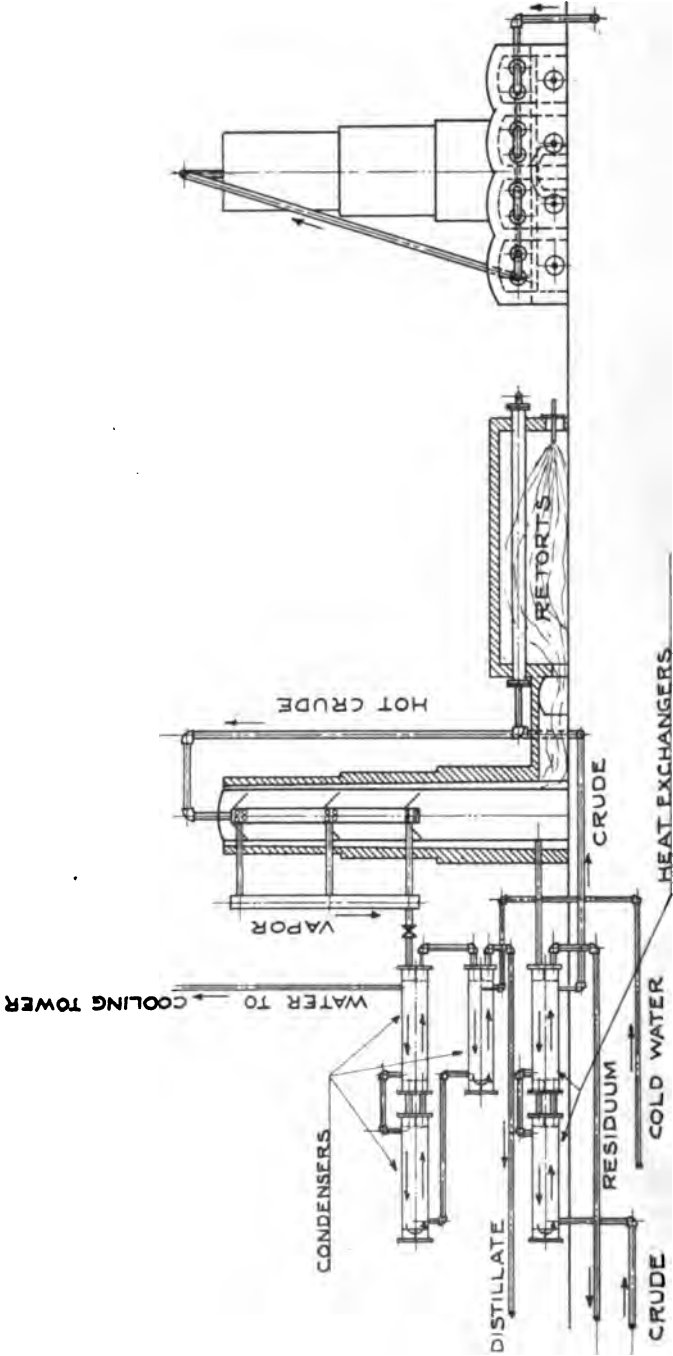


FIG. 13.—SMALL TRUMBULL TOPPING PLANT, PACIFIC CRUDE OIL Co., MIDWAY FIELD, KERN COUNTY, CALIFORNIA.

The General Petroleum Co., in adopting the Trumbull system, purchased a half interest in the Trumbull patents which to date are as follows: No. 996736, patented July 5, 1911; No. 1002474, patented



FIG. 14.—TRUMBULL TOPPING PLANT OF PACIFIC CRUDE OIL CO.

Sept. 5, 1911; No. 1070301, patented Aug. 12, 1913; also patents secured in 72 foreign countries.



FIG. 15.—FRONT VIEW OF TRUMBULL PLANT OF GENERAL PETROLEUM CO.

The special features claimed for the Trumbull process are:
The saving of heat in the evaporating chamber by means of conical

spreaders in the evaporators, all oil being forced to run down the inside surface of the shell, which is heated on the outside by the escaping flue gases.

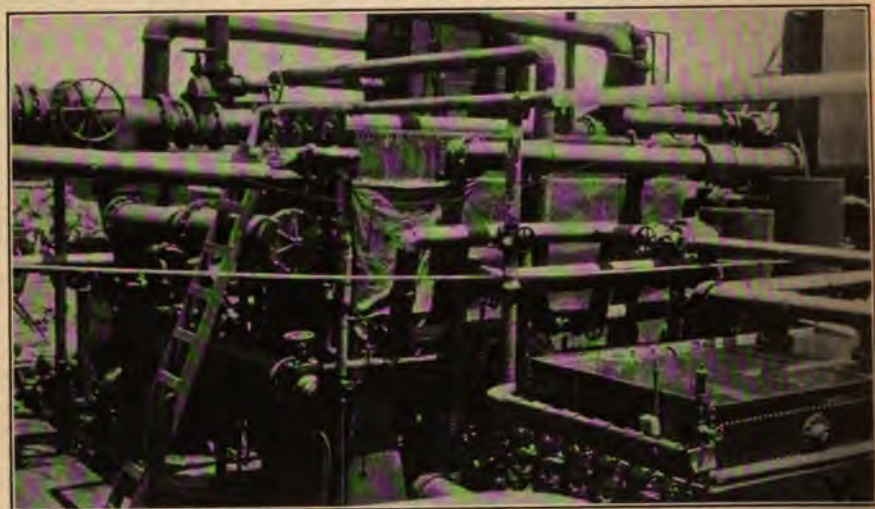


FIG. 16.—SEPARATORS.

The condensing of the first-run vapors and the re-evaporation of these distillates by passing them through a specially designed separator, which derives its heat from the outflowing first-run vapors.



FIG. 17.—RETORT.

And the recovery of the re-evaporated distillates from separate compartments of the separator thereby furnishing different gravities of finished products.

The Vernon plant was completed in 1912, at a cost of about \$80,000 and has been in continuous successful operation since that time. The plant consists of two independent batteries, each battery having two retorts discharging into a single evaporating chamber.* Each battery has two separator boxes, where the condensed distillates are re-run into finished products; also three heat exchangers about 4 ft. in diameter with 18-ft. tubes; two first-run vapor condensers of approximately the same size; and four re-run vapor condensers 30 in. in diameter with 18-ft. tubes.

In addition to the condensers, there are water-jacketed coolers consisting of one or two small pipes laid within a larger pipe for the purpose of further cooling the distillates before going to the storage tank.

In this plant, tubular condensers and water-jacketed pipe coolers have



FIG. 18.—VAPOR CONDENSERS AND PIPE JACKETED COOLERS.

been used in every case instead of the ordinary condensers generally used about a refinery.

Since the plant was built, it has been found advisable to make several changes and additions; one important change being in the retorts. The original retort consisted of six 12½-in. pipes about 20 ft. long connected together at their ends with 4-in. pipe returns. Vent pipes were inserted at one end of each retort to lead off gases which, if trapped in the large

* This chamber is called a "separator" on all the plants described except in connection with the Trumbull plants where it is called an evaporating chamber.

pipes, would cause unequal heating and sagging of the retorts. It was found that there was not heating surface enough, the escaping gases in the stack running as high as 1,200° F. Additional 4-in. retort pipes were, therefore, placed above the 12½-in. pipes, thereby increasing the heating surface about 50 per cent., the result being that the temperature of the escaping gases was reduced to about 650° F. (Fig. 17.)

The company has lately torn down one of the retorts and rebuilt it entirely of 4-in. pipe. This change has resulted in a marked improvement.

Another important feature added to the original construction is a large rectangular dephlegmator and re-evaporator erected at each unit through which all the first-run vapors pass. This dephlegmator, it is claimed, has improved the quality of the products by removing the heavier distillates which originally discolored them.

The oils handled average 20° to 21° Bé.; the residuum manufactured, 16° to 17° Bé.; the tops manufactured, No. 1 of both 55° and 58° Bé., No. 2 of 45° Bé. and still bottoms of 35° to 37° Bé.

Special cuts of various gravities are also turned out to order.

*Temperatures Maintained at Plant and Percentages of
Oil and Water*

Crude run to heat exchangers.....	80° F.
Heat exchangers to retorts.....	264° F.
Out of retorts.....	445° F.
Residuum to storage.....	190° F.
Water in oil, per cent.....	0.5
Fuel consumed, per cent.....	1.6

The last item does not include oil consumed in pumping about the refinery, this being done from the main pipe-line pumping station, and by auxiliary electric pumps. The above, however, includes pumping of circulating water through the cooling towers.

The course of the oil through a single battery of the Trumbull plant at Vernon is as follows:

The crude, previously heated by passing through heat exchangers *J* and *H* (Fig. 19), is forced through the retort pipes. From there, it passes upward by a single pipe to the top of the evaporator *A*, and is sprayed over a conical spreader which directs the fluid against the inner surface of the shell. In its downward course, this heated crude comes in contact with additional conical spreaders, continuously forcing it against the heated shell. The vapors separate from the hot crude in the evaporating chamber and enter a vertical pipe which is perforated directly under the apex of each conical spreader. This pipe delivers the vapors into three outlets which again terminate on the outside in a single down pipe and lead the vapors through the upper separator *B* to the three inner chambers *L* of the recently constructed dephlegmator *D*. The inner

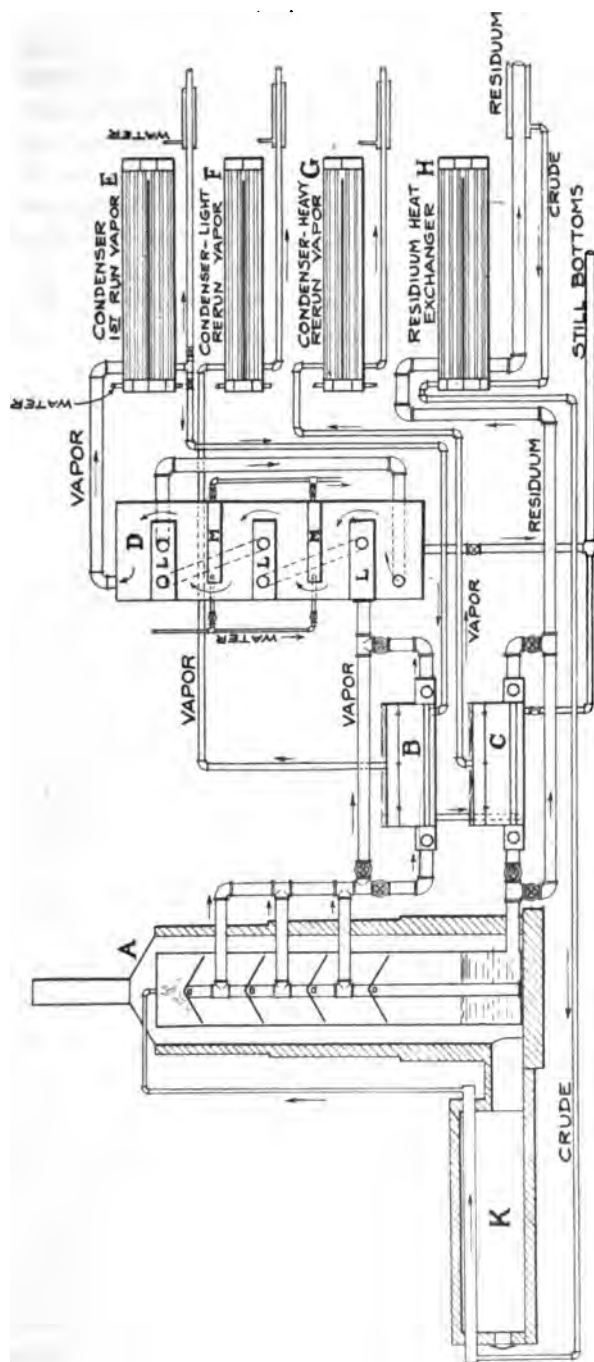
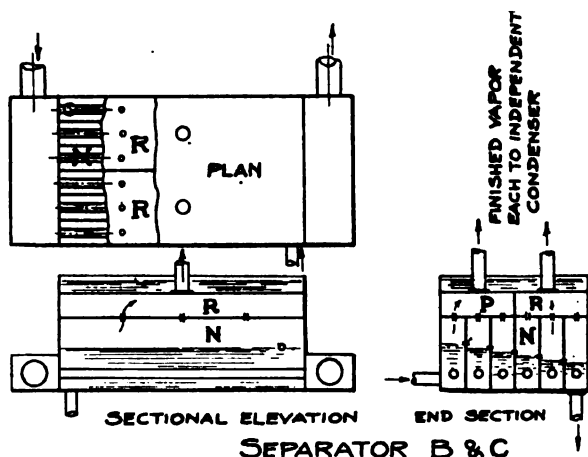


FIG. 19.—GENERAL PETROLEUM CO., TRUMBULL TOPPING PLANT, VERNON, LOS ANGELES COUNTY, CALIFORNIA.

chambers *L* are sealed from the rest of the interior of the rectangular box forming the dephlegmator.

Between each of the three chambers *L*, are two water compartments *M* placed across the box. The incoming vapors enter the lower chamber *L*, pass from the lower to the middle chamber *L*, and from the middle into the upper chamber *L*, and from there through a pipe into the open portion at the bottom of dephlegmator *D*. These gases coming in contact with chambers *L*, which are filled with the hotter vapors, are re-heated.

As the re-heated vapors flow up in the main box *D*, they come in contact with the cooling surfaces of the water compartment *M*, which condense and throw down the heavier distillates on to the heated surface of the chamber *L*, the latter evaporating any of the lighter oil, which might be entrained with the condensed heavier distillates. The result of



SEPARATOR B & C

FIG. 19A.

this operation, it is claimed, is that all the heavy discolored distillates finally reach the bottom of the dephlegmator, while the light clean vapors pass onward and out of the dephlegmator to a battery of two large tubular water condensers *E*, which are of similar construction to the residuum heat exchangers.

The distillate from condenser *E* flows to the separator *B* entering the bottom compartment at one corner, traveling forward and backward six times through a series of compartments *N*, heated by the hot vapors coming from the evaporator *A*. (Fig. 19 A). As the distillates flow through these heated compartments *N*, they are evaporated, the vapors rising through openings into either of the two upper compartments *P* and *R*, from each of which the vapors are led to two independent condensers *F*.

The outside upper portions of the separators *B* and *C* are water jacketed, and as the hot vapors rise in the compartments *P* and *R*, any heavy

distillate carried by the vapors is condensed and drops back into compartment *N*, so that only the lighter vapors escape from the respective compartments.

The unevaporated distillates pass on to separator *C*, where they are treated in the same manner as in separator *B*, the vapors being conducted to two independent condensers *G*.

On account of the distillates being heavier in separator *C* than in separator *B*, a greater heat is required to evaporate them, which is derived from the hotter residuum instead of from vapors. This arrangement of two separators with four independent compartments gives four separate re-run distillates of different gravities, practically without additional cost for fuel, the heat expended in doing this work being derived from



FIG. 20.—TRUMBULL PLANT OF THE SHELL CO.

the original heat absorbed in the retorts, except for the steam which is admitted into the separators to help drive over the vapors.

The unevaporated distillate from separator *C* passes on with the heavy distillate from dephlegmator *D* to the storage tanks.

Bypasses are arranged in both the vapor and residuum lines, so that all or only a portion of the vapor or residuum may travel through the separators; by this means, only the required heat is obtained.

The residuum after falling to the bottom of evaporator *A*, passes through the separator *C*, and on to a series of three heat exchangers, each exchanger having a diaphragm in the shell, which guides the oil backward and forward; it then enters the next shell and so on into the third shell from which it passes through water-jacketed pipe coolers to storage.

The cold crude, entering these exchangers through the tube compart-

ment at the end of the shell and by means of three diaphragms, is guided backward and forward through the tubes four times, this process being repeated through each of the exchangers, the crude entering the coldest exchanger and passing out of the hottest.

THE SHELL CO., OF CALIFORNIA

The Shell Company, through an affiliated company known as the Simplex Refining Co., purchased in May, 1915, all the Trumbull topping system patents, at a reported figure of \$1,000,000, and is constructing at Martinez a Trumbull topping plant in connection with a complete refinery.

This plant will be supplied by an 8-in. pipe line running from the Coal-inga field to Martinez, where the company has rail and deep-water facilities.

The topping plant now being constructed is a single unit and has an economical capacity of about 5,000 bbl. of crude oil in 24 hr. No money is being spared to make it as complete and economical as the process will permit. In general design, the plant is similar to the General Petroleum Co.'s Vernon plant. Many improvements have been made; one of the most noticeable is the spreading of the different parts so that they can easily be inspected and repaired; the change in design of the retorts, and various minor changes which will be to the advantage of the new plant.

Another feature of the Shell plant is the addition of an evaporator column in place of the dephlegmator lately erected at the Vernon plant.

Besides the main topping plant, the Shell company has erected a smaller Trumbull retort with its evaporating tower for the purpose of running asphaltum and doing other experimental work. This part of the plant is not complete and is not otherwise referred to in this paper.

Like the Vernon plant, all condensing and cooling will be done in tubular condensers.

The process will be carried on as follows:

The cold crude enters first the three vapor condensers *E* and *U* (Fig. 21), then passes forward and backward four times through the tubes of each exchanger, then to the coldest of the residuum heat exchangers *H*, through the five exchangers *H* in the same manner as the other exchangers, leaving the hottest exchanger to enter the top of the evaporator column *T*, which is practically of the same construction as the main evaporator except that the heat is derived from a surrounding vapor jacket. The crude on entering the column *T* is diverted to the heated side in its downward course by the conical deflectors and finally drops into the base of the column. From there, it passes through the retort *K*, then up to the top of the evaporator *A*, where in its downward course, it is again deflected against the shell heated by the furnace gases, and falls to the bottom of the tower.

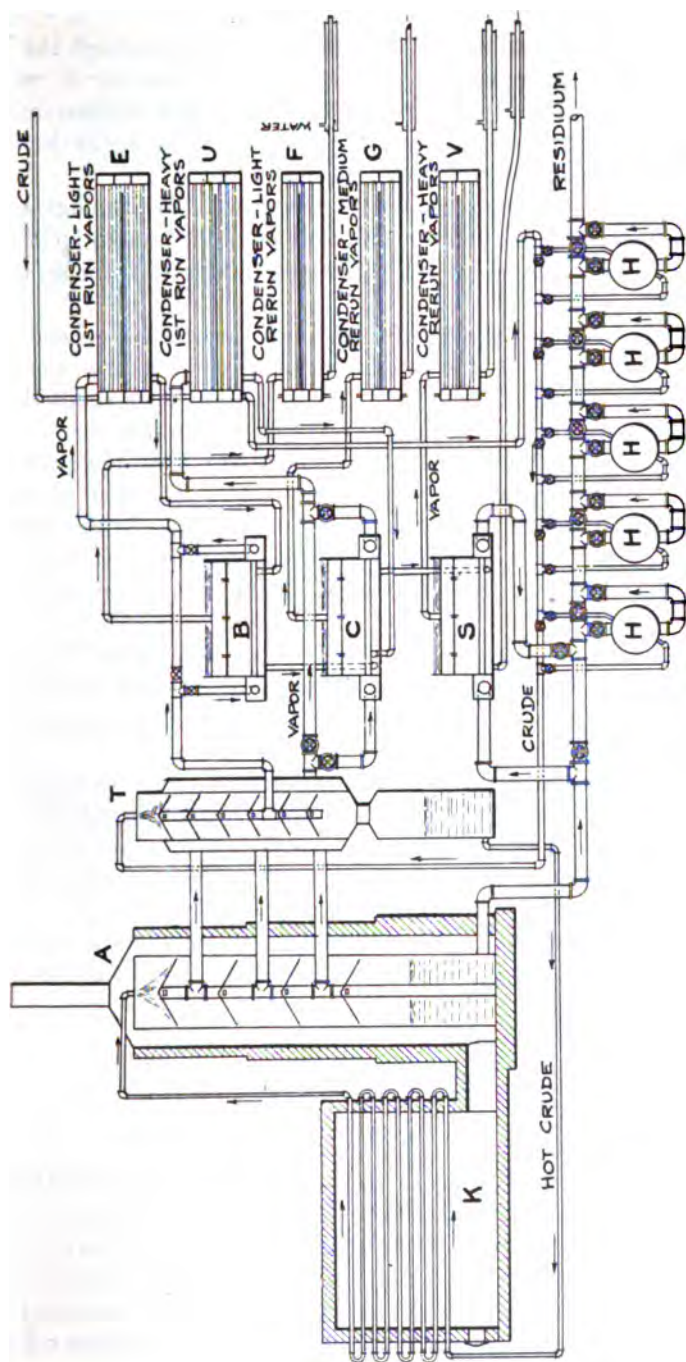


FIG. 21.—SHELL COMPANY OF CALIFORNIA, TRUMBULL TOPPING PLANT, MARTINEZ, CONTRA COSTA COUNTY, CALIFORNIA.

The first vapors to come off are from the hot crude in the evaporator column *T*, which are led to the separator *B*, passing through the heating tubes and out of the opposite end, then to the condenser *E*, where the distillates are condensed and returned to the lower compartment (*N*) of separator² *B*, the process of evaporation being the same as described in the General Petroleum Co.'s Trumbull plant.

The vapors on leaving separator *B*, which will have the lightest gravity, are condensed in condenser *F* and then pass off through jacketed pipe coolers to storage, while the unevaporated distillates from separator *B* pass on to separator *C*.

The second vapors coming off from evaporator *A* are collected in the central perforated pipe and led to the jacket of the evaporator column *T*, after imparting a certain amount of heat to the inner shell, they pass through the heating tubes in separator *C*, then to condenser *U*, the condensed distillates being returned to compartment *N* of separator *C* to be evaporated as previously described. The vapors then flow to condenser *G*. These distillates are of a heavier grade than those from separator *B*. The unevaporated distillates flow from separator *C* to separator *S*, which is heated by the hot residuum passing through it on its way to the exchanger.

The vapors from separator *S* pass off to the condenser *V* in the same manner as those from *B* and *C*; they are heavier than those from the other two separators. The unevaporated distillate from separator *S* is not further refined and is run to storage.

Each of the three separators has two separate vapor chambers, each having a separate outlet connected to its independent condenser so that the three separators will give six different cuts of varying gravity of re-run products. An additional product will be the still bottoms drawn off from the separator *S*.

The Shell Co. is designing an additional Trumbull plant of equal capacity to be erected alongside of the present one, in which still further improvements are contemplated.

ASSOCIATED OIL CO.

Plant at Gaviota, Santa Barbara County

The Associated Oil Co. has a topping plant of about 5,000 bbl. daily capacity at Gaviota, Santa Barbara County, which consists of two systems; one a battery of ordinary stills and the other a battery of pipe retorts. As I have not dealt in this paper with ordinary stills, but have confined myself entirely to the other types, and as the Associated Oil Co. is building a much larger pipe retort topping plant at its Avon refinery, of

² For detail drawing of separator see Fig. 19A.

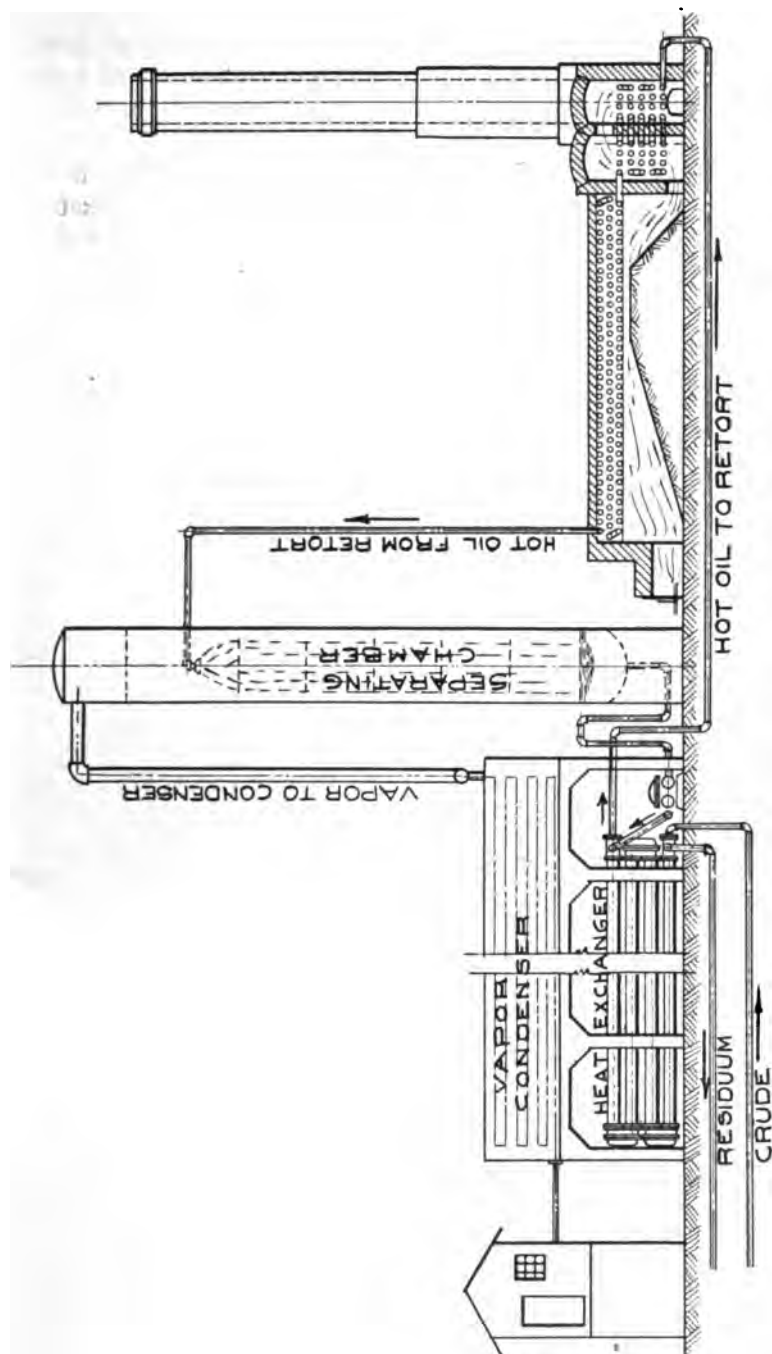


FIG. 22.—ASSOCIATED OIL CO. TOPPING PLANT, AVON REFINERY.

the same general design as the one at Gaviota, I will not describe either of the Gaviota systems but will describe the topping plant now being constructed at the Associated Oil Company's new refinery at Avon.

Plant at Avon, Contra Costa County

A retort topping plant is being erected as an addition to the existing refinery and consists of a flat square construction, divided into three separate furnaces. The direction of all flame is at right angles to the pipes. Placed above the three furnaces will be three rows each of 34 4-in. pipes, about 20 ft. in length with flanged return bends at either end. At the rear end of the furnaces is an enlarged flue which contains 40 4-in. pipes for the purpose of absorbing as much heat from the flue gas as



FIG. 23.—VIEW SHOWING FRONT OF FURNACES UNDER CONSTRUCTION.

is practicable. By constructing three long furnaces, over which the pipes are placed at right angles, the pipes can be supported at four points. Also, short cast-iron T-bars are placed between each row of pipes over the division furnace walls, which support the upper rows and prevent bending of the pipe due to the action of heat. The roof over the furnaces, which is covered by a double thickness of hollow tile with a layer of gravel, rests on the upper row of pipes.

The main outside walls of the furnace come just inside of the return bends at either end of the pipe. There is also a light auxiliary wall placed on the outside of the return bends. The space between the main wall and this auxiliary wall is covered on top with short light sheet-iron plates which prevent a circulation of cooling air on the return bends. The whole construction admits of removing any pipe or joint without loss of time.

In connection with the retort is a separating tower about 5 ft. in diameter and 40 ft. high, into which the heated residuum enters at a point about 10 ft. from the top. The residuum strikes a succession of baffles as it falls to the bottom of the separator; the light vapors flow upward and out of the vent pipe at the top of the separator.

The heat exchanger consists of a nest of 24 10-in. pipes about 40 ft. long mounted on a steel frame, the 10-in. pipes being fitted with return bends, and having five 2-in. pipes passing through each from end to end with connections so arranged at either end of the 10-in. pipes that the heated residuum begins its flow at one end of the 10-in. pipes and flows backward and forward through each successive 10-in. pipe until it has reached the end; in the opposite direction the cold crude oil is admitted into the 2-in. pipes at the cool end of the exchanger and leaves the exchanger at its hottest end.

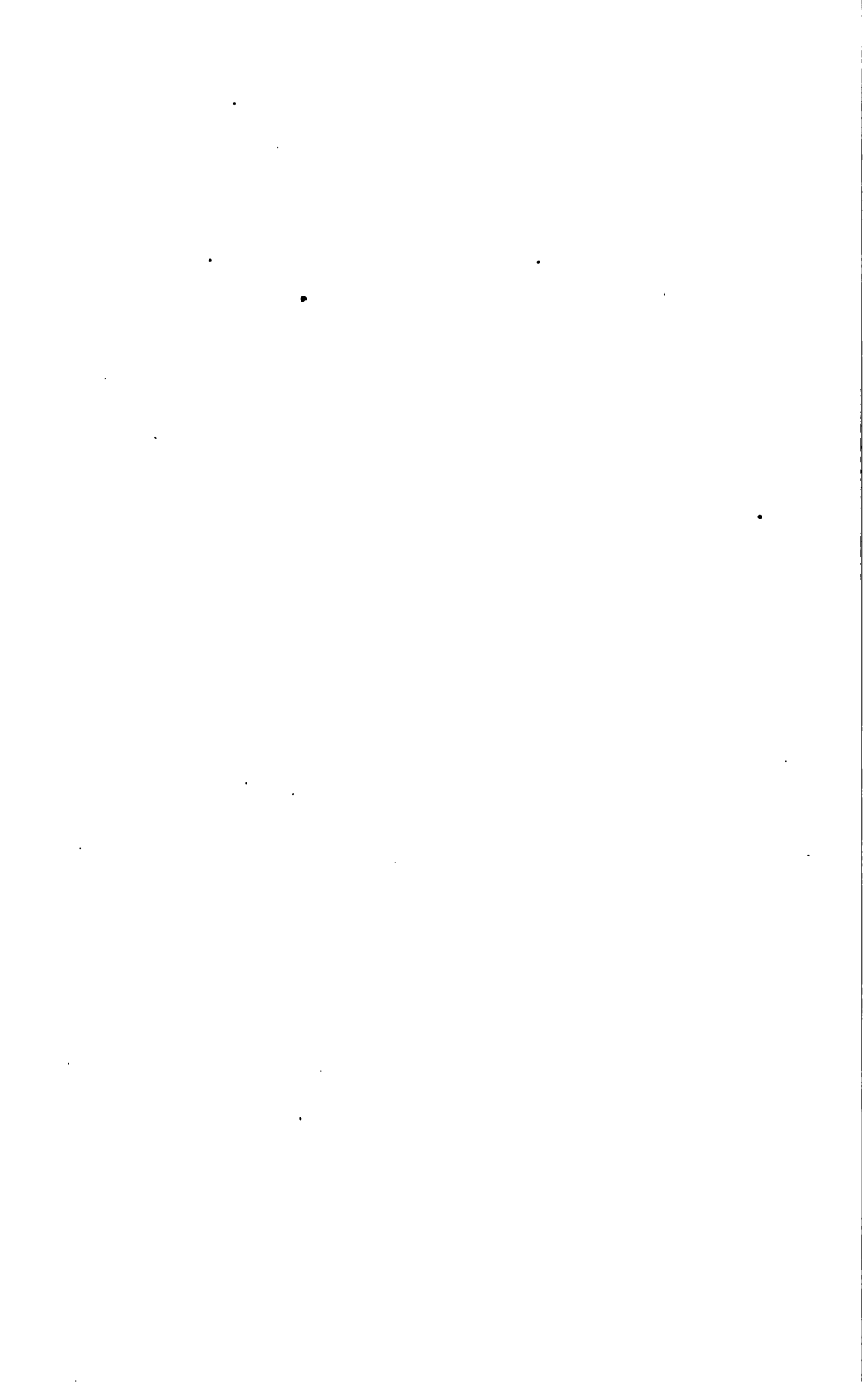
An ordinary water-cooled condenser for condensing the vapors from the separator is used in connection with the retort.

In operating the plant, the crude oil enters the exchanger, passes out of the hottest end and enters the retort, at a pressure of about 50 lb., through the lower corner pipe of the flue chamber, traverses backward and forward through the eight lower pipes in this chamber, then flows successively through each row upwards to the top; from there it traverses backward and forward over the bottom row of pipes lying over the furnace until it reaches the front, then in the same manner through the second and third rows back to the front of the furnace; from there it is led, by a pipe with a pressure regulating valve in it, to the upper portion of the separator where the spraying takes place. The residuum is then pumped from the bottom of the separator through the heat exchanger to the storage tanks. The vapors from the separator are led direct to the condenser.

The residuum will be delivered from the retorts to the separator at about 450° F. As this particular topping plant has not yet been completed, I cannot give definite data as to its operation, but the same design of exchanger used in connection with ordinary stills topping oil in the same plant, receives the residuum at 450° F.; the crude oil enters the exchanger at 70°, leaves it at 260°, and enters the stills at approximately that temperature.

The estimated economical capacity of this plant with clean oil will be about 5,000 bbl. of crude each 24 hr., taking off about 15 per cent. of 50° Bé. tops.

In closing this paper, I wish to express my appreciation of the courtesies extended to me by the officers and employees of all the companies herein referred to, as it would have been impossible to compile so much accurate data without their hearty cooperation.



DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

The Geology of the Iron-Ore Deposits In and Near Daiquiri, Cuba*

BY JAMES F. KEMP,† NEW YORK, N. Y.

(San Francisco Meeting, September, 1915)

THE iron ores of southeastern Cuba present a subject of exceptional geological interest. Their relations to the inclosing rocks are varied and in some cases unusual. The problem of their geological origin is not a simple one and for this reason has occasioned some divergence of views among previous observers. Profiting, however, by these earlier studies and by more detailed field and laboratory observations as well as by the experience of added years of mining, much more satisfactory descriptions and interpretations can now be prepared.

PREVIOUS WORK

In 1884, interest in the southeastern coast of Cuba became active among the consumers of iron ore along the Atlantic seaboard of the United States. Active development was undertaken by the Juragua Iron Co., Ltd., a corporation affiliated with the Bethlehem Steel Co., and the Pennsylvania Steel Co. The property of the Juragua company, which is situated about 15 miles east of Santiago, was studied in June and July, 1884, by Dr. James P. Kimball, at the time Professor of Economic Geology in Lehigh University and one of the best known of American mining geologists. Dr. Kimball published on his return two papers, the first important descriptions which we have of the local geology.¹ It is

* In the preparation of the present paper the writer has had the cordial support and interest of Charles F. Rand, President of the Spanish-American Iron Co. For this aid the fullest acknowledgment is here gladly made.

† Professor of Geology, Columbia University.

¹ James P. Kimball. Geological Relations of the Specular Iron-Ores of Santiago de Cuba, *American Journal of Science*, vol. xxviii, third series, pp. 416 to 429 (1884). The Iron Ore Range of the Santiago District of Cuba, *Trans.* xiii, 613 to 634 (1884-85). To the combined reprints of the above papers as distributed to his friends, Dr. Kimball added reproductions of three maps which accompanied his report to his clients. The first is of the East and West Mines near Firmeza, on a scale of about 150 ft. to the inch and with contours at 3-m. intervals. Some notes on the geological formations are written on the map. The second contains three separate figures, viz.: (1) an outline map of the coastal district from the Bay of Santiago to the Rio Bucano; (2) a geological section through the Juragua Hills to the Gran Piedra; (3) a claim map of the surroundings of Firmeza. The last plate contained a series of geological sections and panoramic sketches of the mines and orebodies near Firmeza.

indeed true that 28 years earlier, D. T. Ansted,² an eminent English geologist, had recorded an account of the copper deposits of El Cobre, west of Santiago, and of others near Cienfuegos. From the earlier contribution one may glean a few details of the local geology at these two mines, but to Dr. Kimball we owe the first easily accessible descriptions of the region to the east of Santiago.

Dr. Kimball describes a syenite as the basal and oldest rock along the coast. He saw its outcroppings in a belt near the sea and learned of its supposed reappearance in the higher portions of the mountain range which culminates in the Gran Piedra. He therefore inferred that it existed beneath the diorites which form the country rock of the intervening belt and which contain the iron ores. Near the sea Dr. Kimball observed the coral reefs which had been obviously raised above the water in recent geological time and which stood at three elevations, respectively 14 ft., 175 ft., and 350 ft. above tide. Farther back the diorite was believed to cover the syenite, through which in some places it may be seen coming up in dikes. The diorite was believed to be thinnest toward the coast and to reach a maximum of about 2,000 ft. farther back. Dr. Kimball inferred that the diorite welled up as an outbreak of igneous rock through the syenite and that from sources of supply back toward the Sierra Maestra it flowed southward toward the sea. The flood of igneous rock was either checked by the salt water or failed in amount and so covered no greater area than approximately that which we now observe. In its course the diorite was supposed to overwhelm reefs of coral limestone, perched on the syenite and still older than those now visible. The effect of the diorite was to turn these older reefs into the crystalline marble which may sometimes be seen in association with the orebodies. Subsequently the circulating rain-waters, acting on the diorites, which contain much iron oxide, took the iron oxides into solution and encountering the ancient coral reefs, were relieved of their dissolved burden so as to replace the limestone with the ore. Dr. Kimball observed structures still preserved in the ore which suggested the organic growth of coral.³

These observations are of great interest, but so far as the origin of the ore is concerned are chiefly of value in bringing out the probable precipitation of some of the ore, perhaps indeed of most of it, through the

² D. T. Ansted. The Copper Lode of Santiago in Cuba, *Quarterly Journal of the Geological Society of London*, vol. xii, pp. 144 to 153 (1856). Extended quotations from this paper are given by Hayes, Vaughan, and Spencer, *Report on a Geological Reconnaissance of Cuba*, pp. 44 to 50 (1901).

On the San Fernando Copper Lodes near Cienfuegos in Cuba, *Idem*, vol. xiii, pp. 240 to 242 (1857).

³ The writer has also observed similar cases at Sigua and has been reminded of the supposed fossil coral of the pre-Cambrian rocks of Quebec, the *Eozoon Canadense*, which was an object of so much study in former years, but which is no longer regarded as of organic origin.

agency of limestone. The precipitating limestone was not an old coral reef perched on the syenite, but a member of an important sedimentary series which is probably the oldest visible rock in the region.

The next paper upon the district is by F. F. Chisholm,⁴ who specially examined the Berraco group some six years after Dr. Kimball's studies, and who also visited the Juragua company's mines. Mr. Chisholm reviews the relations of the syenite and diorite, and notes the three terraces of recently uplifted coral. With the view, however, that the ore has replaced ancient coral reefs he dissents entirely. Much better exposures had been prepared at Firmeza in the Union mine during six years of work. A vertical body 200 ft. high and of goodly although not definitely stated width had been exposed. Quite naturally Mr. Chisholm found difficulty in believing that it was an old coral reef replaced with ore. The slabs of limestone observed in the diorite near the ore were interpreted as fragments torn off by the diorite in its upward journey and derived from some older formation. The great vertical orebody at the Union mine was inclosed in diorite. It had a wavy band of chloritic clay on the east side of the ore, with small parallel, lenticular streaks of vitreous quartz. Outside the clay was 2 ft. of sand, loosely compacted and colored by manganese, but displaying a sharp contact against the diorite. The other openings at Juragua were similar but presented less clearly marked relations. At Berraco, by way of contrast, no limestone was exposed. Mr. Chisholm finally interpreted the ore as a direct concentration in an igneous dike, which was originally characterized by a high percentage of iron; or else it is a distinct band forming a portion of a larger dike. The source of the ore would therefore be below, and the ore itself would continue below the effects of atmospheric action.

Two years after Mr. Chisholm's visit, an examination of the Sigua group together with visits at Juragua and Daiquiri was made by the late George W. Goetz of Milwaukee, whose observations were communicated to the late Professor Hermann Wedding of Berlin, partly by letter and partly through short articles in the press. Professor Wedding assembled the data in a paper although not himself personally familiar with the mines.⁵ The oldest rock is stated to be the syenite of earlier writers, upon which rests an oolitic, coralline limestone belonging to the Jurassic and of the horizon of Quenstedt's Beta, of the upper White Jura. The limestone is much disturbed and together with the syenite is pierced by outbreaks of diorite, which have changed the limestones neighboring to them into white marble. The red hematite is associated with the diorite in the most intimate manner as in most localities the world

⁴ F. F. Chisholm. Iron-ore Beds at the Province of Santiago, Cuba. *Proceedings of the Colorado Scientific Society*, vol. iii, p. 259 (1888-90).

⁵ H. Wedding. Die Eisenerze der Insel Cuba, *Stahl und Eisen*, vol. xii, No. 12, pp. 545 to 550 (June 15, 1892).

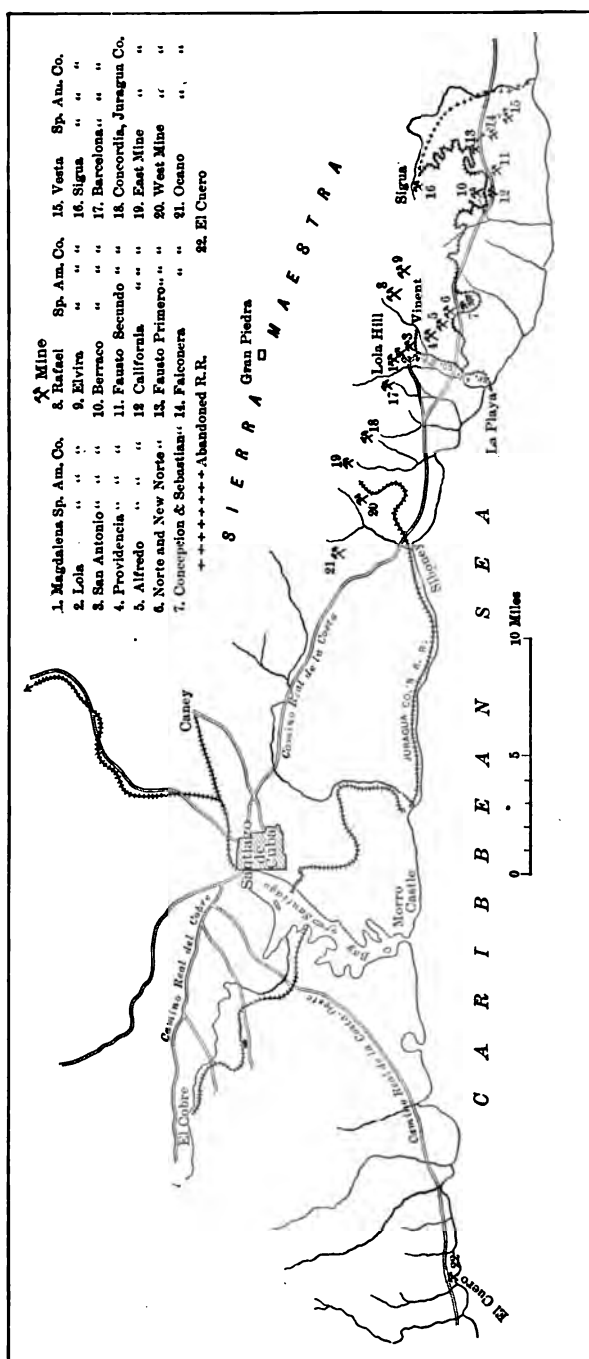


FIG. 1.—SKETCH MAP OF THE MINES EAST AND WEST OF SANTIAGO DE CUBA.

over, such as the Lahn district in Germany, and the Lake Superior district in North America. West from Santiago Devonian sandstone and Silurian limestone constitute the stratified rocks. Dr. Wedding emphasizes the great abundance of surface blocks of ore in the iron districts east of Santiago, and thinks it probable on the analogies with other districts that the iron ores form beds rather than veins. The exposures described as veins may be folded beds. The diorite is often porphyritic and the iron ore is at times involved with chlorite and epidote; and frequently appears simply as a replacement (pseudomorphose) of limestone. Many details of the engineering features follow.

In the interesting work, *Cuba and Porto Rico, with the Other Islands of the West Indies* by Robert T. Hill,⁶ a very brief sketch of the geology of Cuba appears on pp. 40 and 41, together with a very small geological map after one by de Castro and Salterain. Limestones of Eocene-Oligocene age constitute the coast east and west of Santiago, with an intrusive mass of granitoid rock at Daiquiri.

The district received careful study from Dr. A. C. Spencer in connection with the reconnaissance of Cuba carried out under General Leonard Wood's direction, during the American occupation in 1901.⁷ Dr. Spencer gives an excellent historical sketch of the development of the enterprises and, regarding the engineering features, quotes extensively from a report by Charles M. Dobson, E. M., to the Military Government, a report which must have been made shortly before the date of his own report. Dr. Spencer determined the syenite of earlier writers to be a coarse quartz-diorite, whose exposures are well shown from Siboney to the Playa, the port of the Daiquiri district. It is covered, in the hills near the shore, by a thin veneer of coral rock. The quartz-diorite (sometimes called merely diorite by Dr. Spencer) extends about three miles inland and is succeeded by a belt or zone of intrusive masses of porphyry. "Near the southern edge of this zone the porphyry occurs in extremely irregular dikes and these dikes become more numerous in the higher foothills to the north. It is within these foothills that the iron ore occurs. Above them the slopes of the Sierra Maestra are composed of bedded volcanic rocks with a few intercalated beds of limestone or limestone breccia. The lower flows are of a basaltic nature, but some of the uppermost are rhyolitic, while upon the top of the mountain the massive rocks are overlaid by rhyolitic breccias, which constitute the base of a series of volcanic rocks occupying the northern slope of the range. These features are indicated in the accompanying diagram Fig. 2. (Fig. 9 of Spencer) which represents also the general geology

⁶ The Century Co., New York (1898).

⁷ C. W. Hayes, T. W. Vaughan and A. C. Spencer. *Report on a Geological Reconnaissance of Cuba*, with a geological section from the coast to the summit of the Sierra Maestra and four small sections of the orebodies, pp. 69 to 82 (1901).

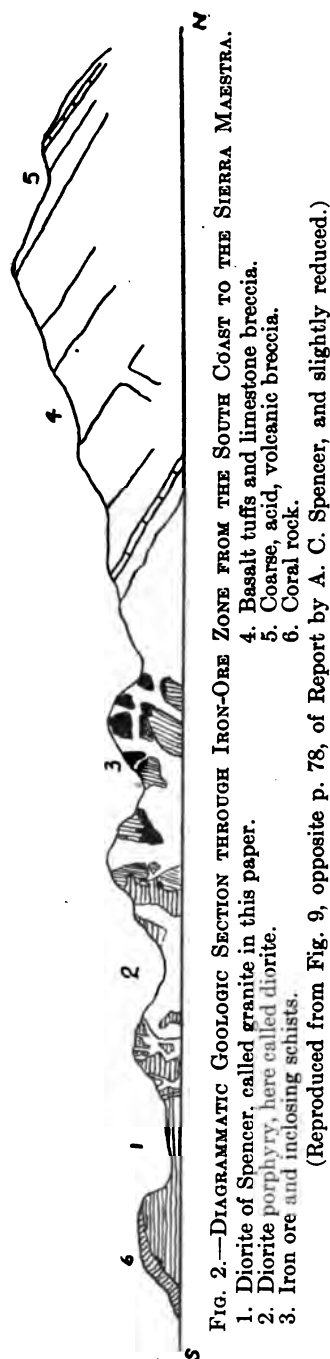


FIG. 2.—DIAGRAMMATIC GEOLOGIC SECTION THROUGH IRON-ORE ZONE FROM THE SOUTH COAST TO THE SIERRA MAESTRA.

1. Diorite of Spencer, called granite in this paper.

2. Diorite porphyry, here called diorite.

3. Iron ore and inclosing schists.

(Reproduced from Fig. 9, opposite p. 78, of Report by A. C. Spencer, and slightly reduced.)

and structure of the Sierra Maestra from Cabo Cruz to the vicinity of Guantanamo Bay."

The volcanic rocks are not intimately associated with the iron ores so that the chief interest attaches to the porphyry, which is the main wall rock of the ore. Dr. Spencer was impressed by the association of hornblende schist as well as crystalline limestone with the ore as exposed in the mines of the Juragua Company, but not much in evidence in the workings on Lola Hill of the Spanish-American company. He was led by these observations and by others elsewhere in Cuba to infer the existence of an old metamorphic series, even older than the serpentine so abundant in some parts of the island. The outbreaks of the quartz-diorite and the porphyry preceded the Oligocene epoch of the Tertiary period, since younger beds on the northern slope of the Sierra Maestra contain fossils of this time. After discussing, at too great length to be cited verbatim here, the derivation of the ores by possible magmatic processes in the igneous mass, and by replacement processes of the included schists and limestones, Dr. Spencer favors the view that the ores were original members of the schist and limestone series and that they have been torn off with varying amounts of their old associated strata and floated upward as included masses in the porphyry. The great slabs or masses of ore are often cracked apart and separated by dikes of porphyry which are regarded as the same rock with the general inclosing walls.⁸ Several cross-sections assembled and reproduced here in Figs. 3, 4, 5 and 6, on a reduced scale are given in support of the view. One of

⁸ In *Trans.*, xlii, 167, 168 (1911), two panoramic views appear accompanying some notes incidental to the Canal Zone excursion.

these with a quotation appears also in J. E. Spurr's *Geology Applied to Mining*, p. 118 (1907) where it is more generally accessible than in the original paper.

In reading Dr. Spencer's description one must bear in mind that the



FIG. 3.

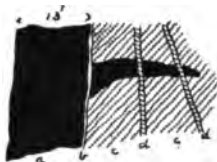


FIG. 4.

FIG. 3.—MAGNETITE AND EPIDOTE NEAR OREBODY IN EAST MINE, FIRMEZA.

h, Hornblende schist; *ep*, granular epidote.

(Reproduced from Fig. 10, p. 80, of Dr. Spencer's Report.)

FIG. 4.—SECTION THROUGH SMALL OREBODY IN NORTHEAST MINE OF JURAGUA COMPANY.

a, Magnetite; *b*, selvage impregnated with chalcocopyrite; *c*, white marble belonging to the schist series; *d*, fine-grained porphyry.

(Reproduced from Fig. 11, p. 80, of Dr. Spencer's Report.)

rock called syenite by earlier writers is named by him quartz-diorite or simply diorite; whereas the rock called diorite by earlier writers and appearing as the wall rock of the ores is called by him porphyry. Dr. Spencer's quartz-diorite or diorite must not be confounded with the diorite of the others.



FIG. 5.

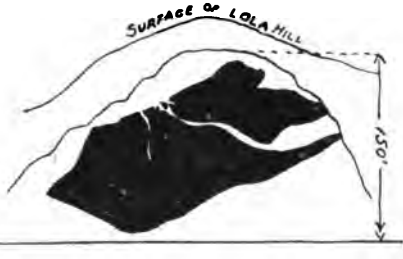


FIG. 6.

FIG. 5 (FIG. 12, *idem*).—WORKING FACE OF LOLA MINE, SHOWING GENERAL RELATIONS OF OREBODY TO INCLOSING PORPHYRY.

FIG. 6 (FIG. 13, *idem*).—GENERALIZED VERTICAL PROJECTION OF MAGDALENA OREBODY AT DAIQUIRI TO ILLUSTRATE INTRUSION OF IGNEOUS ROCK.

So far as the sedimentary rocks and especially the limestones are concerned, Dr. Kimball regarded them as coral reefs older than the modern reefs, but although no definite geological epoch is assigned to them, apparently in his view they grew in conditions not greatly changed from those of today except in the matter of subsidence. Professor Hill

quoting two Spanish geologists refers them to the Eocene-Oligocene, but Dr. Spencer believes them older than the Oligocene because on the north side of the Sierra Maestra, Oligocene fossiliferous strata rest upon the same types of eruptives as appear on the south side and are later. Mr. Goetz must have supplied Professor Wedding with some sort of palaeontological proof for making so close a determination of them as a subdivision of one portion of the Jurassic. Fossils would be of extreme interest. One cannot well avoid suspecting that they are a continuation of the thick and tilted limestones which one traverses along the wonderful mountain highway, which climbs the escarpment northwest of Santiago to 2,000 ft. above the sea. The winding road crosses in true Alpine manner a great section of well stratified limestones with a northerly dip of 30 degrees or more.

In detailed areal work in the Daiquiri region as soon as one leaves the artificial exposures of the mines and railway cuts, or the freshly eroded ledges of the brooks, the difficulties of the tropics assert themselves. The weathering is so severe and the advance of decomposition so rapid, that fresh rock is often hard to find. Kaolinized and chloritized specimens do not always give a satisfactory indication of the nature of the eruptive rocks. By the aid, however, of E. W. Kohl, Jr., the engineer of the Spanish-American Iron Co., and his assistant, Alfred H. Weaver, a carefully located and quite complete illustrative series of rock specimens, over 800 in number, were collected from the lands of the Spanish-American Iron Co. The general geology has been worked out with the aid of these in the areas not actually visited. Slides for microscopic study have been freely prepared. The writer has also visited the larger mines of the Juragua company and has been further aided by a collection of specimens of ores and rocks made in 1913 by C. R. Corning, to whom acknowledgments are due.

GEOGRAPHICAL RÉSUMÉ

In connection with the iron ores we are deeply concerned with a portion of Cuba which lies east of Santiago along the coast of the Caribbean Sea. The iron mines are limited to a belt which extends not over four or five miles back from the shore. The mines have not as yet been developed more than 20 miles east of Santiago Bay nor nearer than five miles to it. They do not come down to the shore but are first found about a mile and a half from the salt water. They are thus limited to an east and west belt two to three miles wide. The ores are in an exceedingly steep series of hills whose slopes at the larger mines approximate 35 degrees, the angle of repose for loose materials. Some smaller openings are at elevations less than 500 ft. above tide, but the larger ore bodies outcropped originally up to and slightly above 1,000 ft., and cut through

the high, steep hills or ridges, in actual or practically vertical position. The smaller orebodies are of all sorts of shapes and are irregular in their distribution.

To the west of Santiago several enterprises have been started, of which at present only the one at El Cuero, eight miles from Santiago Bay is active. Its ores lie quite near the shore and are reported as similar in geological relations to those on the east.

Thirty-seven miles westward from the mouth of Santiago Bay is Guama, with a port called Chirivico. The Cuban Steel Ore Co. made extensive developments on some iron-ore lands about 14 years ago, but the enterprise is not now active.

OUTLINE OF THE GEOLOGY AT DAIQUIRI

The oldest geological formation on the lands of the Spanish-American Iron Co., so far as our observations go, is of sedimentary character. Its chief representative today is limestone, but quartzites have also been discovered and at Sigua black slates are to be seen in close association with the limestone. This series is cut up into small irregular exposures by the extensive development of igneous rocks which have greatly metamorphosed it, and have produced contact zones of marked perfection and scientific interest.

The limestone can be seen in the Barcelona claim in a not appreciably metamorphosed condition. It is then a dense blue variety, strongly reminiscent of the Cambrian and Ordovician limestones of the Appalachian area of the United States, and the great Cretaceous limestones of northeastern Mexico. Satisfactory dip and strike were not easy to secure on account of the massive character and the neighboring igneous rock. One reading on the Barcelona claim was N 55 W., 53 W. The limestone is again abundant to the north of Vinent, and to the northeast toward Sigua. It has been found on the south at various points even so far to the east as the Coco River. While we have no positive evidence that all the exposures belong to the same formation, yet in the absence of the evidence of fossils, no one of which has been yet discovered, we may assume that they do.

Quartzites have been discovered to the east of La Playa, the port of the Daiquiri district, and extend nearly to the Coco River in an east and west belt from a half mile to a mile and a half back from the shore. They are chiefly quartz sandstones of pronounced fragmental character but show the effects of metamorphism. Sericite is recognizable and apparently zoisite. There are particles of feldspar. Fig. 7 will illustrate them.

In the pits at Sigua a fine dark-green slate was found, which under the microscope is a greenish, almost isotropic mass with a few grains of quartz. It was interpreted as a fine feldspathic tuff, now kaolinized and stained green with chlorite.

There are other fragmental rocks both near the mines at Sigua and along the trail to the southeast. They were once feldspathic in composition but are now so greatly kaolinized as not to be more sharply determined. They seemed to be an old volcanic breccia. The strike and dip some two or three miles southeast of Sigua were N. 20 E., 10 W. We have no evidence of the relations of this rock to the limestones and quartzites.

It is possible that in part or as a whole these sediments may represent the ancient metamorphic series mentioned by Dr. Spencer. But no hornblende schist has been found away from the mines, and such ap-



FIG. 7.—QUARTZITE EAST OF LA PLAYA.



FIG. 8.—GRANITE (QUARTZ DIORITE), BERRACO R.R.

In FIG. 7 and in all the subsequent reproductions of thin sections (except Fig. 43), the actual field is 0.1 in. or 2.5 mm.

parently schistose representatives as were observed in the Spanish-American company's workings were lime-silicates, obviously the results of contact metamorphism. Abundant evidence of faulting movements and of shearing under pressure are to be noted both in the open cuts of Lola Hill and near Firmeza. The limestone-quartzite series does not seem to be so much metamorphosed as would be implied in Dr. Spencer's description of the ancient schists.

Granite.—The review of the earlier papers has shown that there has been a general disposition to regard the syenite or quartz-diorite as the oldest rock in the region. As the writer is anxious to avoid all confusion of this rock with the diorites which are associated with the ore, it will be called granite. The ordinary field worker would unquestionably collect it as granite. When examined under the microscope it always has abundant quartz, making syenite an undesirable name. Sometimes the feldspar is predominantly orthoclase; sometimes, and more often, it is oligoclase. Biotite and hornblende and augite are all three to be seen in

one slide or another but biotite is most common. The usual zircons, titanites and magnetites appear and at times there is a marked tendency to develop micro-pegmatites. The rock is rather coarsely granitoid in texture and of light or actually white color. Dr. Spencer's description of the rock as quartz-diorite coincides perfectly with most of the thin sections examined by the writer. These are, however, phases well over toward granite, which name, both from ordinary field usage and to avoid confusion is here employed. We know the granite to be later than the limestones because it changes the latter to beautiful white marble and in places has produced contact zones of great perfection. Along the Berraco R. R., where it skirts the east side of the San Sebastian claim there is one contact zone of coarsely crystalline epidote right between the granite and the white marble. The ore of the San Sebastian is in a garnet zone produced by the granite from the limestone; while that of the neighboring Concepcion claim is in epidote produced in the same way. There can be no question that the granite is intrusive in the limestones and is later in age. But the granite itself is at times abundantly provided with angular inclusions of a much darker diorite. In the cuts along the Berraco railway along the east side of the San Sebastian claim, the ledges of granite display a good many of these angular fragments and lead one to believe in some older unidentified member of the composition of diorite. On the other hand, and as all the previous observers have mentioned, dikes of diorite cut across the granite and are generally believed to be the supply conduits for the great masses of diorite which contain the ore. Yet there are dikes in the mines which are also later than the diorite including the ore and which may be of the same group as the dikes that are to be seen cutting the granite. No positive evidence has been obtained by the writer to show that the granite is older than the ore-bearing diorite. The included fragments which appear in the granite may have been derived from the latter rock, unless the earlier observations are destructive of this view. As will be later shown granitic, pegmatitic and even dacite-porphyry dikes cut the orebodies in one mine and another. Should they be offshoots from the deeper-seated granite, the latter might even be later than the main orebodies themselves. The included, angular fragments would thus correspond to the older diorite which contains the orebodies.

The granite is best developed east and west of La Playa, from which it extends nearly to Vinent. It is also seen in good exposures along the railway from Firmeza to Siboney. Undoubtedly it extends some miles along the coast, Siboney to La Playa and beyond each.

Diorite.—In the neighborhood of Vinent the granite yields to a rather coarsely crystalline diorite which extends some distance to the north and is widespread east and west. The diorite is sometimes visibly porphyritic from phenocrysts of feldspar and for this reason was probably

called porphyry by Dr. Spencer. On careful microscopic study the writer is unable to draw any very sharp distinctions as against the wall rocks of the orebodies and therefore is disposed to regard this great dioritic intrusive as essentially one geologic unit.

The diorite is a gray rock, which may assume a light color on weathering, but which, in the varieties rich in hornblende may be a very dark gray. In some exposures it is of moderately coarse grain, comparable with an average granite having components up to 0.2 in. or 5 mm.; again it is much finer. The most abundant mineral is plagioclase, well twinned and affording extinctions on either side of the plane of twinning from a few degrees to as high as 25. Varieties in the andesine and labradorite series are thus chiefly represented with an occasional member as basic as bytownite. In the coarser varieties the plagioclase has the usual habit of the components of the granitoid rocks. The crystals are roughly as long as broad and of irregular outline. In the more finely crystalline phases there is a marked tendency of the plagioclase to become rectangular and even at times square, reminding one of the characters of the dike rocks. Yet the specimens were certainly gathered from the large intrusive mass as distinguished from the relatively narrow dikes. The plagioclases are rarely zonal. Untwinned feldspar is only now and then to be seen, so that orthoclase is a minor component. Quartz is not uncommon, but it is only a subordinate member. The feldspar may be four-fifths the slide as a maximum and one-half as a minimum. The hornblende is the common green variety in almost all the slides, but it shows frequent indications of having once been brown and to have bleached in alteration. One constantly suspects common green hornblende of having been derived from augite, but the probabilities are very strong that in the Daiquiri rocks the hornblende is an original mineral. In the coarser diorites it is irregular in outline but in the finer-grained varieties in which the plagioclase becomes rectangular, the hornblende exhibits excellent prismatic development. Hexagonal sections and elongated ones with parallel sides are the rule. The hornblende may make up as much as one-third the section; it may sink to a fifth. Irregular magnetite is freely associated with the hornblende, but seldom appears in the feldspar. Apatite is often seen and is even included in the magnetite, being thus an older mineral. Pyrite is now and then an associate of the magnetite.

In some slides and even those in which alteration has not made particularly great progress, calcite may nevertheless be detected. It is secondary and has undoubtedly been yielded by hornblende and feldspar. From this outline, which may be considered the normal, variations occasionally appear. Biotite in a visibly bleached or discolored form is not infrequent as a minor associate of hornblende, but it has never been observed in more than a subordinate rôle. In an abandoned cut in the southern workings of the old El Norte mine a specimen of diorite was

gathered which has much colorless augite associated with the hornblende. The plagioclase at the same time becomes basic. Its extinctions reach 25 degrees with the twinning plane. Epidote has also begun to develop.



FIG. 9.

FIG. 9.—DIORITE, LOWER TUNNEL, EL NORTE MINE.

The mineral with parallel bands is plagioclase. The clear white is quartz. The dark mineral is chiefly hornblende with a little magnetite. Crossed nicols.



FIG. 10.

FIG. 10.—DIORITE FROM EAST WALL OF EAST MINE, FIRMEZA.

The slide is chiefly plagioclase. Hornblende is recognizable in the upper middle part by its cleavage. The very dark minerals on the right lower side are extinguished plagioclase. The rock is practically identical with the one illustrated in Fig. 9.

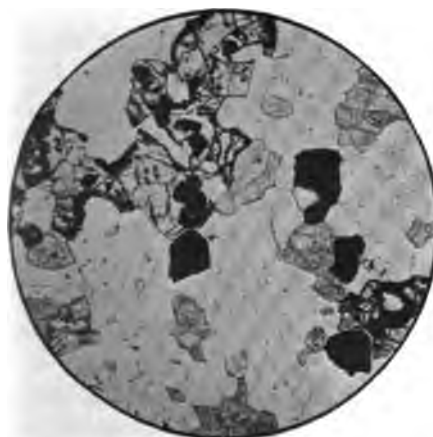


FIG. 11.

FIG. 11.—OLIVINE GABBRO IN WHITE LIGHT.

The mineral with high relief is olivine with dark, included magnetite. The gray mineral of less relief is augite. The white mineral is plagioclase. The black mineral is magnetite.



FIG. 12.

FIG. 12.—THE SAME AS FIG. 11, BUT TAKEN WITH CROSSED NICOLS.

The rock was thus half diorite, half gabbro in its characters, but it seemed to be only a variation from the usual diorite and not a separate intrusion.

Olivine-Gabbro.—A still more marked departure from the usual diorite is found on the Santa Rosa claim, on the southwest extension of Providencia Hill. Excellent fresh rock was gathered on the surface in an area of noticeable but moderate magnetic attraction. When the rock was examined microscopically it proved to be a typical olivine-gabbro, Figs. 11 and 12. Plagioclase, pale-green augite, pale-green olivine, and unusually coarse bits of magnetite are its components. The magnetite develops curious skeleton growths. A very little brownish-green hornblende appears in parallel growths with the augite. It may be that the gabbro is a separate and distinct mass from the ordinary diorite which contains on the north side of Providencia Hill the new Providencia mine and on the south side the Alfredo. Forest growth and concealed exposures made the close tracing of relationships difficult, if at all practicable. The rocks also are not easy to identify sharply without constant appeal to the microscope. Olivine-gabbro makes one apprehensive of titanium in any ore appearing where it constitutes the wall rock.

Wall-Rocks of the Juragua Mines.—The same diorite as has been described above from the Spanish-American company's lands appears as the chief wall rock of the Juragua company's East Mine and West Mne. In the slide from the former as shown in part, in the photomicrograph, Fig. 10, plagioclase, a little orthoclase, hornblende, a little biotite and one small augite crystal together with magnetite and a little pyrite were observed. In the latter, in a more finely crystalline rock, were plagioclase, hornblende, a little quartz and much magnetite.

Inclusions in the Diorite.—North of Vinent along the Daiquiri River and also on the slopes of Providencia Hill, the diorite is richly provided with angular inclusions of the same dense, dark diorite which we find in the granite, as earlier described. The component minerals of the inclusions are markedly well bounded. Rectangular feldspars, prismatic, sharply six-sided hornblendes and bits of magnetite chiefly constitute them. No pronounced contrasts in mineralogy can be detected as against the usual diorite, but there is more hornblende and the texture is different from the coarser varieties. Apparently the inclusions represent some older solidified rock.

In a recent valuable paper on the Boulder Batholith of Montana, Paul Billingsley⁹ has remarked the abundance of dark, angular dioritic inclusions in this famous granite. He has suggested that they represent an older and first chilled basic shell, afterward broken up and included in the later arising granite from a new propulsive effort. The contrasts of the inclusions are so great at Daiquiri, especially with the granite, and diorite is such a common rock, the writer leans to considering the fragments as derived from an older mass, broken up by a newer intrusion.

⁹ *Bulletin* No. 97, Jan., 1915, p. 42.

Alteration of Diorite near the Ore.—Near the orebodies the diorites are noticeably altered and become very unsatisfactory for microscopic study. Epidote appears in secondary veinlets and sometimes in larger veins. Next these veins of epidote the diorite becomes dense and greenish gray, passing practically into a felsite. The process generally summed up under the word propylitization has taken place on a large scale. Dark silicates yield chlorite which stains the whole mass of the rock green. Feldspars become muddy and kaolinized and finally a mass of secondary products must be studied in which one faintly traces the outlines and structures of once fresh crystals. So constant is this change near the orebodies that one cannot avoid the conviction that thermal waters have been actively circulating.

White Siliceous Rocks.—In all the larger mines, dikes or at least longitudinal belts of white feldspathic and sometimes quartzose rock are visible, at times in close association with the orebodies and on one side. Lee Reifsnider, the superintendent of the mines on Lola Hill, remarked of this light-colored rock that in passing from diorite into it he looked for ore on the far side; but in passing from ore into it, he anticipated barren rock beyond. In the field it was difficult to decide on the nature of these rocks. They seemed in most instances silicified diorite and were considered to be the result of some thermal action associated with the ore formation. This explanation is not impossible. A specimen was gathered in the Lola-San Antonio workings which under the microscope proved to be a shattered quartz vein with some isotropic mineral filling the cracks, Fig. 13.

A second specimen collected in the Magdalena mine is a siliceous dike of granite or quartz-diorite, akin to pegmatites. Quartz, acidic plagioclase, sometimes in micro-pegmatitic intergrowths, and a little epidote made up the slide, Fig. 14. A third specimen from the East Mine of the Juragua company is a beautiful case of a micro-pegmatitic granite dike and is illustrated in Fig. 15. A fourth case from the east side of the West Mine of the same company appears to be an old sandstone. The first three cases were believed to be in some way associated with the ore formation, since the acidic rock seemed often to form one wall of the ore. The fourth case, if correctly interpreted must be older than the ore. But besides all these instances there is a fifth on the west side of the West Mine at Firmeza, where an undoubted granitic or pegmatitic dike with an offshoot cuts, or in earlier exposures did cut, the ore. It is a richly quartzose rock with quartz in acidic plagioclase, so as often to suggest micropegmatite, Fig. 16. A little hornblende and magnetite are also discernible. In the small branch of the larger dike, biotite is the dark silicate. Of these rocks, the sandstone, if correctly interpreted, stands by itself. The first three certainly seemed to have some close relationships with the ore. They might conceivably be

later, but the field relations did not suggest it. At the Concordia Mine a white, feldspathic rock was also found associated with the ore. Under the microscope it is chiefly a finely crystalline feldspathic ag-



FIG. 13.

FIG. 13.—SHATTERED QUARTZ VEIN, LOLA MINE.



FIG. 14.

FIG. 14.—VERY ACIDIC QUARTZ DIORITE, CONSISTING OF QUARTZ, THE MOST ABUNDANT MINERAL IN CLEAR, WHITE, LIGHT OR DARK GRAY AREAS; MICROPEGMATITE, THE MOTTLED AREAS; AND EPIDOTE, THE DARK SHREDS. MAGDALENA MINE.



FIG. 15.

FIG. 15.—MICRO-PEGMATITE FROM A DIKE IN THE EAST MINE, FIRMEZA.



FIG. 16.

FIG. 16.—ACIDIC DIKE, WEST MINE, FIRMEZA.
Acid plagioclase with micro-pegmatitic quartz on the right. Fine-grained diorite, forming the wall rock on the left.

gregate, reminding one of nothing so much as somewhat badly altered trachytes or bostonite dikes. Blotches of less definite igneous textures appear as well and make the rock very puzzling for sharp determination. Probably the original diorite has been severely affected by thermal

waters, has lost its dark silicates and has changed over to this feldspathic residue.

In the New El Norte mine of the Daiquiri group a light-colored porphyritic dike with branches penetrates the ore in Cut 3 as illustrated

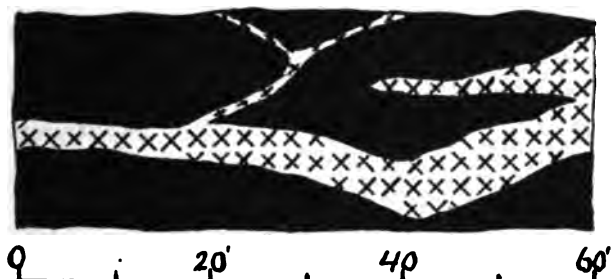


FIG. 17.—RAMIFYING DIKE OR THIN SILL OF DACITE-PORPHYRY CUTTING THE ORE ON CUT 3, NEW NORTE MINE.

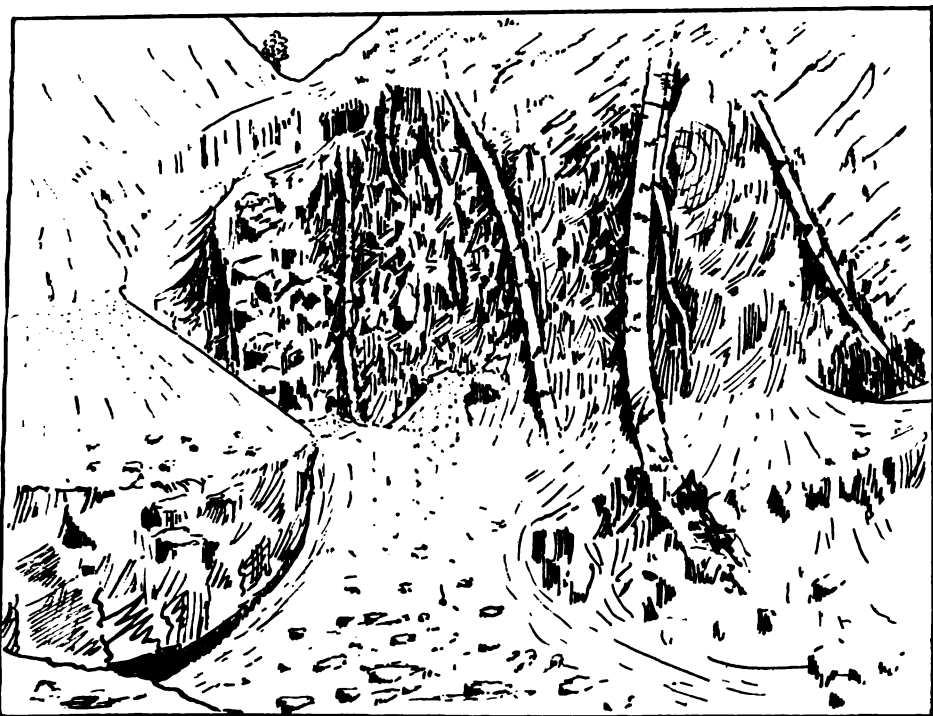


FIG. 18.—NARROW DIORITIC DIKES, CUTTING ORE IN WEST MINE, FIRMEZA.
 Sketched from a Photograph taken by W. L. Cummings.

in Fig. 17. It is clearly later than the ore and proves under the microscope to be a dacite porphyry. Phenocrysts of plagioclase, rarer orthoclase, and quartz are set in a finely crystalline groundmass of quartz and

feldspar with microgranitic texture. Tracts of biotite may be determined in the surviving chlorite.

Diorite Dikes.—In the larger mines, such as those on Lola Hill, the main supply of the Spanish-American company, and in the East and West Mines of the Juragua company, as well as in some of the smaller openings, narrow black dikes cut wall rocks and ore alike. They are undoubtedly later than both and have entered long after the deposition of the ore. Since the best and freshest ones collected cut also the light-colored feldspathic rock at the West Mine of the Juragua company, the



FIG. 19.—FINE-GRAINED DIORITE FROM A NARROW DIKE, WEST MINE, FIRMEZA, AND SHOWING CHARACTERISTIC TEXTURE.

The rods and squares are plagioclase, which constitutes the larger part of the rock. The dark minerals are hornblende and magnetite.

narrow dikes are believed to be later than the white rocks. There seem to be two sets of dark dikes. In Stope 6 of the underground work in the Magdalena mine a later one crosses an earlier one. Six from the Lola Hill have been examined in thin section.

One variety is a very feldspathic diorite porphyry which cuts the ore of the Magdalena mine. Phenocrysts of zonal plagioclase are set in a groundmass of smaller plagioclases, all pretty sharply bounded by their own faces. The phenocrysts reach 2.5 mm. but the groundmass is made up of components from a quarter to a tenth this size. A few six-sided

chlorite pseudomorphs after hornblende prisms remain. A little magnetite is dotted through and there is some secondary calcite.

The narrower dikes from Lola Hill have much smaller components of which plagioclase is the chief. The plagioclase shows a marked tendency to develop rectangular or square cross-sections. It is a basic variety. The dark silicates are more abundant than in the diorite porphyry just mentioned. Presumably the common one was hornblende but only chlorite survives to indicate it. The dikes have much fine magnetite. Calcite and epidote are also often in the slides.

An unusually fresh dike from the West Mine, Juragua, was 1 ft. thick. It contained, as shown in Fig. 19, beautifully fresh plagioclase which affords rectangular sections. Irregular bits and shreds of leek-green hornblende in less amount than the plagioclases are scattered through the slide, as are fine bits of magnetite. The slide reminds one strongly of diabases, with their lath-shaped plagioclases and irregular augites, but there seems no good reason to regard the hornblende as secondary after augite.

These dark, basic dikes undoubtedly represent the last igneous rocks to enter. They are closely akin to the chief wall rocks of the ore. They are sometimes associated with copper minerals along their borders, so that green copper stains now appear near them and in mining the iron ore it has been possible to throw to one side a few tons, rich enough in copper to send to El Cobre near Santiago. Usually, however, only a few green stains are the result.

SUMMARY OF ROCKS ASSOCIATED WITH THE ORE

The general associates of the ore are now before us. They constitute a complex and in some respects a very puzzling group to arrange in a logical series and to place in their time relations. One cannot help associating the granitic or pegmatitic dikes with some large parent body. The natural one is the intrusive granite mentioned at the outset. Yet this granite has produced contact zones on the older limestone with orebodies, whereas the granitic and quartz-porphyry dikes are, in two cases at least, later than the large orebodies. We can only suspect the possible connection without being able to prove it. If true, it would move the Playa granite into a place later in time than the main ore deposition or at least place its expiring effects and offshoots at this late period.

The great mass of diorite certainly preceded the main ore formation, unless one concludes with Dr. Spencer that the masses of ore have been torn off from older formations below and floated upward to their present positions. Details of the ore now to be given cast much doubt upon this hypothesis. Finally many basic dikes and some acidic ones followed the ore formation.

THE ORE

The ore is mainly a fine-grained magnetite. Dr. Spencer noted this character although almost all the earlier descriptions speak of it as specular hematite. A belief prevails at the mines that it is a fine aggregate of both



FIG. 20.



FIG. 21.

FIG. 20.—POLISHED SLAB OF ORE FROM LOLA HILL, BY DIRECTLY REFLECTED LIGHT. The light-gray areas, showing the structure of lath-shaped crystals of plagioclase, are magnetite. The irregular dark patches are quartz and garnet.

FIG. 21.—NEARLY THE SAME VIEW AS FIG. 20, BUT WITH OBLIQUE ILLUMINATION SO AS TO BRING OUT THE QUARTZ IN THE DEPRESSIONS.



FIG. 22.



FIG. 23.

FIG. 22.—ANOTHER AREA WITH PYRITE STANDING OUT IN RELIEF FROM THE MAGNETITE.

FIG. 23.—A STILL DIFFERENT AREA WITH PYRITE IN RELIEF.

specular hematite and magnetite, and that analysis has shown this to be the case. Some light might be thrown upon the question by determining the ferrous iron and its relations to the ferric, but care would of necessity

be exercised against pyrite which is widespread in small amounts. A representative suite of specimens taken at intervals from one end of the Lola Hill mines to the other has been tried with the magnet. All but one or two were immediately attracted by it. One or two were inert. We are forced to conclude that the greater part of the ore is magnetite. In the contact zones, as represented in the smaller mines, rosettes of specular ore are not infrequently to be seen.

In a polished slab of ore, illustrated in Figs. 20 to 23 inclusive, the structure is a cellular one with quartz, garnet and pyrite in the interstices of the magnetite. The magnetite itself looks like nothing so much

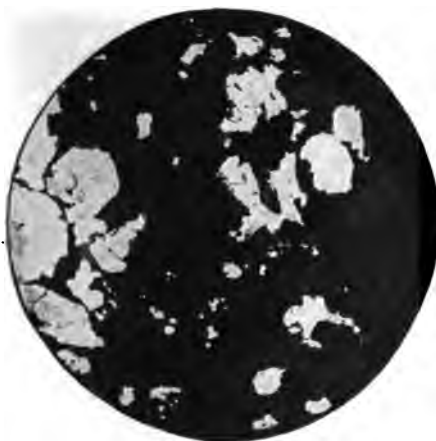


FIG. 24.

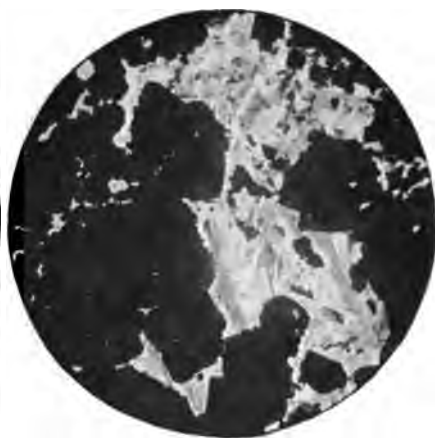


FIG. 25.

FIG. 24.—QUARTZ INTERGROWN WITH MAGNETITE. MAGDALENA MINE.
FIG. 25.—QUARTZ INTERGROWN WITH MAGNETITE IN ORE FROM FIRMEZA.

as a replacement of lath-shaped crystals of feldspar such as we often see in the diorite itself. One can hardly resist the impression that in the slab some original rock texture is preserved. This texture is most easily explained as due to a replacement of the diorite with ore.

The ore from the portions of the mines above the water level is open textured and cellular because of the oxidation and removal of pyrite, which in the deeper workings appears in unaltered condition. On Lola Hill in the northern workings in the Magdalena claim one notes little in the way of admixture in the ore except pyrite, which increases with depth. But as we pass to the south and into the Lola and San Antonio workings we find notable and significant amounts of quartz. The quartz does not seem to be a secondary and late introduction into cavities in older magnetite, but an original component of the ore. Blades, minute, irregular, angular lumps and dust of magnetite are contained in the quartz. Some magnetite at least apparently crystallized with it (Fig. 24). Similar relations obtain at Firmeza, as shown in Fig. 25.

To a small extent in the ore of the Magdalena mine, but to an increasing degree as one goes south through the Lola into the San Antonio, garnet becomes a component in the ore. Some specimens are a mere mass of garnet and magnetite in blotchy intergrowth. The garnet may even be the chief mineral. Garnet and quartz may both be intergrown and associated in the ore, as shown in Fig. 26. Garnet and calcite are also associated in the ore, Fig. 28. The relations of the garnet and magnetite are so intimate that they must have been produced by the same general process. In one slide from a specimen of ore from the middle of the orebody on a section in Lola Hill, 1,050 ft. from the north end of the



FIG. 26.

FIG. 26.—QUARTZ AND GARNETS INTERGROWN WITH MAGNETITE. MAGDALENA MINE.



FIG. 27.

FIG. 27.—OLIVINE INTERGROWN WITH MAGNETITE. LOLA MINE.

Magdalena, a brightly refracting, rather high-index, optically positive mineral was found in a slide, which is otherwise chiefly garnet and magnetite. The mineral is irregularly cracked but has also a rectangular cleavage and parallel extinction. It appears to be olivine, although the association is unusual for olivine (Fig. 27). Still one is reminded of the olivine found by Charles Palache and described by C. H. Warren¹⁰ in veins in the Cumberland, R. I., titaniferous magnetite, and obviously an after-effect in an igneous mass. The magnetite at times shows much epidote. The epidote does not seem to be necessarily secondary but to belong with the garnet as one of the original crystallizations in the process of ore deposition.

¹⁰ C. H. Warren. Petrography and Mineralogy of Iron Mine Hill, Cumberland, R. I. *American Journal of Science*, 4th ser., vol. xxv, pp. 35, 36 (1908). An earlier case described from the Alps by E. Weinschenk is cited by Dr. Warren. Weinschenk attributes to it a method of origin such as is here supported for the Daiquiri case.

A sample of garnet was collected in the San Antonio mine and was kindly analyzed by R. B. Herr, Jr., chemist of the Spanish-American Iron Co.

	Molecular Ratio			Molecular Ratio	
SiO ₂	35.23	587	CaO	30.22	537
Al ₂ O ₃	0.82	8	S	0.01	...
Fe ₂ O ₃	30.11	198	P ₂ O ₅	0.05	...
FeO	0.71	10	H ₂ O + ...	0.15	...
MnO	0.10	1	CaCO ₃	2.56	...
MgO	trace	...			
			Total	99.85	

When recast for molecular ratios we have: SiO₂, 587; Fe₂O₃ (198) + Al₂O₃ (8), 206; CaO (537) + FeO (10) + MnO (1), 548. The sesquibases are thus too high, or the silica somewhat low and the protoxides very much too low for an exact garnet formula of 3 SiO₂, (FeAl)₂O₃, 3(Ca, Fe, Mn)O, but it is clear that the garnet is predominantly the lime-iron garnet, andradite, most characteristically developed in the contact zones. There was probably some magnetite or hematite in the sample as 2.5 per cent. Fe₂O₃ was soluble in dilute hydrochloric acid.

Along the northeastern side of the San Antonio workings, in the summer of 1914, was an extended streak of garnet rock. Excellent crystallized specimens of a brownish-red variety could at times be obtained where they had projected into a cavity which had later been filled with calcite. This streak or belt of garnet in the diorite seemed attributable to the same general processes as the ore, but it may have been formed from a long included slab or series of slabs of limestone.

Very similar garnet rock may at times be observed in the mines of the Juragua company but the rocks have not been studied in such detail as those at Daiquiri. From the East mine a specimen was gathered which reveals under the microscope pale-green garnet with the ore and quartz mingled. Apparently the quartz was the last mineral to be deposited. In the slide amphibole is sparingly present. From the Concordia mine, garnet rock was gathered which is likewise pale green under the microscope. The garnets are possessed of optical anomalies.

With the ore of some of the smaller mines of the Spanish-American company is associated a black, heavy rock called by the Spanish miners "malo mineral" or "bad ore," and sometimes because of its blue-black color "piedra azul" or blue rock. Specimens were collected both from the New Norte and from the Old Norte mines, and similar occurrences do not fail in other openings. The writer was not always able to distinguish the piedra azul from the ore itself, but on request the miners invariably decided without hesitation. The piedra azul is shown by the microscope to be a phase of the mineralization which led to the ore. The commonest variety is a very fine-grained aggregate of magnetite and

augite, Fig. 29. One would at first take it to be a narrow dike, but in another piece a fine-grained mixture of garnet and pyroxene made up the rock, allying it with the contact effects. The best explanation is therefore that while the mineralizing agents in one place produced ore by replacement of the diorite with magnetite, they may have afforded dense and closely crystalline bodies of magnetite and augite or both these and garnet, close alongside.

One other striking feature of the larger mines of the Spanish-American company should be emphasized before passing to the smaller openings which are based on characteristic contact deposits between intrusives and



FIG. 28.

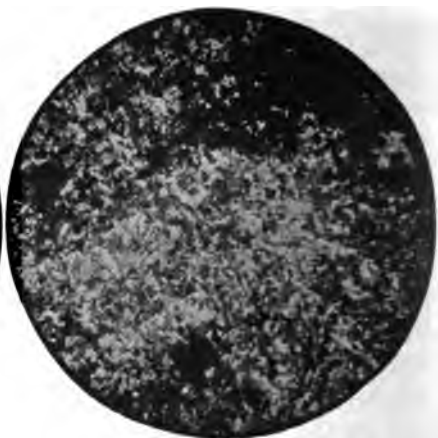


FIG. 29.

FIG. 28.—CALCITE IN MAGNETITE FROM ONE OF THE MINES ON LOLA HILL.
FIG. 29.—PIEDRA AZUL OR MALO MINERAL, A FINE-GRAINED INTERGROWTH OF
MAGNETITE AND AUGITE.

limestones. One cannot study the great open cuts in Lola Hill without being impressed with the evidences of movement and crushing. The diorite is crushed along planes which run in close parallelism with the ore. It is squeezed and sheared around the buttress-like masses of magnetite which have presented a resisting front to the movement, but which probably antedate some though not all of it. The alteration of the diorite is excessive and it has apparently been subject to some influence, especially in the vicinity of the ore which has almost obliterated the hornblende and feldspar in favor of chloritic greenstone. In this crushed zone, traversing Lola Hill like a backbone, the great slab-like masses of ore are distributed along a northwest and southeast line.

Dr. Spencer sketched several cross-sections of the orebodies when at the mines in 1901, which are reproduced in Figs. 3 to 6, along with one from Firmeza. To those in Figs. 3 to 6, is added one more, Fig. 30, reproduced from panoramic photographs of Lola Hill made by the photog-



FIG. 30.—PART OF A PANORAMIC VIEW OF LOLA HILL, REPRODUCED TO ILLUSTRATE THE HUGE, TABULAR MASSES OF ORE.



FIG. 30a.—A SECOND PANORAMA OF LOLA HILL TAKEN AT A DIFFERENT STAGE IN THE OPERATIONS AND SHOWING A DIFFERENT DISTRIBUTION OF OREBODIES FROM FIG. 30.

rapher of the Spanish-American company, Francisco Crego. The sketches only afford cross-sections, but we can justifiably imagine that they continue for relatively long distances at right angles to these cross-sections. They are much deeper than wide and in the aggregate, so far as mining has shown, they are on Lola Hill longer than deep. The ore has been followed for 2,800 ft. from the north extreme of the Magdalena

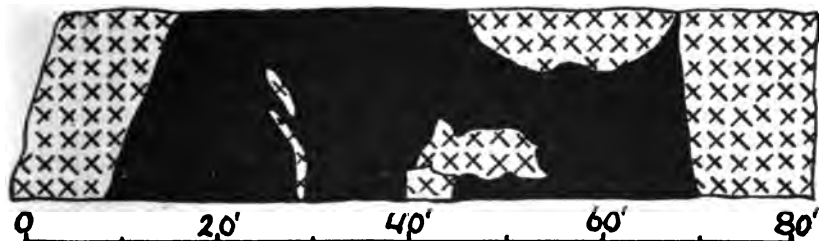


FIG. 31.—SKETCH OF CUT 4, NEW NORTE, TO ILLUSTRATE SURVIVING FRAGMENTS OF DIORITE IN ORE.

mine to the south extreme of the San Antonio. The old summit of the hill with outcropping ore was about 1,000 ft. above tide, while the lowest exploratory tunnel which was cut is 500 or 600 ft. lower.

Thus the whole arrangement of the orebodies is an elongated one, such as would arise from mineralization along a fissured strip. One can hardly imagine such a long and relatively narrow slab or series of slabs to be torn off in depth and floated up edgewise into their present position. On the contrary, the structure of the ore, reproducing the



FIG. 32.—MASS OF DIORITE IN ORE, CUT 3, NEW NORTE.
The diorite appeared to be a surviving fragment from a fault breccia.

interlacing feldspar, the associated pyrite, the quartz and the garnets constantly remind us of replacements by just such emissions as we generally believe have produced the contact metamorphic zones.

Another very striking feature is this: When we prolong the general strike of the orebodies on Lola Hill to the southeast across the valley of the Daiquiri River, we run almost, if not quite exactly, into the Providencia mine on Providencia Hill. Along the same general line over

the top and down the other side of the hill is the Alfredo, prolonged again beyond another valley by the Norte and New Norte. They all line up as well as one could expect along a great belt of crushing and faulting. They display much the same characters from end to end, but the greatest mineralization is on Lola Hill.

Effects of crushing do not fail in the smaller mines. Fig. 31 is reproduced from a sketch made in Cut 4 of the New Norte. Fragments of wall rock remained in the mass of ore. In another part of the hill in Cut 3 of the New Norte, relations shown in Fig. 32 were found. A piece of wall rock had survived in the ore along a zone of crushing.

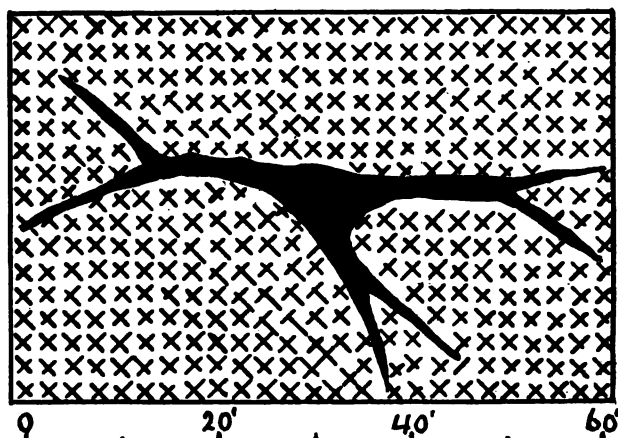


FIG. 33.—RAMIFYING VEIN OF MAGNETITE WHICH APPARENTLY HAD REPLACED THE DIORITE ALONG INTERSECTING FISSURES. ABANDONED WORKINGS OF THE ALFREDO MINE.

One is led to the conclusion that while the diorite mass was still hot in the depths or after it had consolidated and had been penetrated by some other and still hot intrusive in depth, a pronounced northwest and southeast fissured zone was formed, up through which came the emissions, fluid or gaseous, which brought the iron for the ore, the pyrite, the garnet and the epidote; the sulphur for the pyrite; and the silica for the quartz, the garnet and the epidote. The lime required by the garnet and the epidote may have been derived from the plagioclase and hornblende of the diorite, or from included blocks of limestone, or deep-lying limestone, or from several of these sources. The iron probably came up as chloride as has been favored by several writers on contact zones.¹¹ The relations at Lola Hill are on a large scale not very different from the dike-like masses of magnetite in the intrusive rock of the Iron Springs

¹¹ C. K. Leith and E. C. Harder. The Iron Ores of the Iron Springs District, Southern Utah, *Bulletin No. 338, U. S. Geological Survey*, p. 77 (1908). V. Goldschmidt. Contact-zones in the Christiania District, Norway.

District of Utah, but the largest masses are in contact zones reversing the Daiquiri relationships. E. P. Jennings¹² has considered the dike-like masses at the Utah locality to be intrusive igneous rock. The same conception has been very seriously considered by some observers for Daiquiri although it has not been recorded in print. Nevertheless, the writer favors replacement along a fissured belt as having the strongest claims to confidence (Fig. 33). In drawing conclusions everyone should keep before him all the points made regarding the shape, associations, lineal distribution, microscopic texture and mineralogy of the orebodies.

THE DISTINCTIVE CONTACT ZONES

There are a number of smaller mines now largely abandoned which are obvious contact zones. They are all in close association with limestone and igneous rock and bear all the marks of having originated by the action of the latter on the former. The two old workings in the Concepcion and San Sebastian are particularly good examples. The former has two pits, a lower on the north side of the hill and not much above the railroad, and an upper and smaller one on top of the hill farther south. The lower cut is entered through granite. A broad zone of epidote succeeds in which were streaks and irregular bodies of ore. The granite had obviously produced from the limestone a contact zone with epidote as almost the only lime-silicate. Together with epidote, bodies of magnetite also resulted. When studied under the microscope the iron ore is also intimately intergrown with garnet and quartz as shown in Fig. 34. On top of the hill the lime-silicate changes to garnet. No obvious reason appeared for the change, but as the two minerals are so closely akin no great difference of physical conditions was probably involved. The Sebastian pit is said to have the ore in garnet also, but darkness prevented the writer from seeing it, on the trip to these claims. On the east side of the Sebastian claim, however, along the Berraco railroad, one passes through cuts of granite when riding south, and then finds a large knob of coarsely crystalline, white marble, extensively changed in places to coarse, prismatic epidote which is illustrated in Fig. 35. The relations are much the same as at the Concepcion lower pit, but no magnetite was noted. No one at all familiar with contact zones produced by intrusive rocks from limestones would hesitate for a moment over the interpretation of these exposures.

The Barcelona claim with two workings of modest size is a case both of ore in the diorite along an apparent vein and in a small contact zone visibly lying between the limestone and the diorite. One walks along

¹² E. P. Jennings. Origin of the Magnetite Iron-Ores of Iron County, Utah, *Trans.*, xxxv, 338 (1905).



FIG. 34.



FIG. 35.

FIG. 34.—RELATIVELY LARGE GARNETS, SHOWING DOUBLE REFRACTION AND IN A FINE-GRAINED INTERGROWTH OF QUARTZ AND MAGNETITE. CONCEPCION MINE.
FIG. 35.—EPIDOTE ROCK, FROM A CONTACT ZONE ON THE BERRACO R.R., PRODUCED BY GRANITE FROM LIMESTONE.



FIG. 36.



FIG. 37.

FIG. 36.—PLATY GROWTHS OF SPECULAR HEMATITE IN QUARTZ. FAUSTO PRIMERO MINE.
FIG. 37.—THIN PLATES OF SPECULAR HEMATITE, BRANCHING FROM LARGER MASSES OF ORE, AND CONTAINED IN QUARTZ. ESCONDIDO MINE.

an exposure of limestone for 200 or 300 ft. before reaching the small contact zone. No satisfactory dip or strike could be taken, the rock was so broken and disturbed. The limestone was partly white and partly blue being apparently original blue changed in places to white by the neighboring intrusive.

The Fausto Primero claim on the southeastern portion of the tract is another contact zone. The ore is largely specular hematite in stringers and larger masses in actual limestone. The observer is forced to conclude that the iron-bearing solutions have come in through all manner of crevices and have deposited the specular hematite in irregular networks throughout the rock. Much quartz is mingled with the specularite, which forms platy intergrowths with it, as shown in Fig. 36. In other specimens gathered as a mass of green silicates the microscope revealed quartz and some minute acicular mineral, believed from its small extinction to be tremolite.

Again on the Escondido claim in a small pit, lean ore and a heavy rock, the "piedra azul," were associated. The ore in thin section revealed innumerable plates of specularite set in a quartz mosaic as shown in Fig. 37.

On the Elvira claim a large open cut has been driven so as to win the stringers and irregular masses of ore which are set in garnet and in a fibrous amphibole which is practically coarse asbestos. Asbestos as an associate of the ore was found in abundance on the Rafael claim. The Elvira rock was examined in thin sections and revealed yellow garnet, calcite and the asbestos in very fine radiating fibers, Fig. 38.

The most extensive and the most interesting of the contact zones and attendant iron-ore deposits are at Sigua, high up on a steep and picturesque mountain side looking away across a valley north to the Sierra Maestra where there were coffee plantations still being cultivated on the slopes. All stages of changes can be seen from limestone through the characteristic silicates to ore, but beyond several dikes which cross the zones and are later, no good exposures of igneous rocks were noted. The dikes are a very porphyritic rock with abundant phenocrysts of feldspar, but are too badly decomposed for microscopic study. There are also green chlorite rocks which may have once been more basic intrusives. The contact effects begin with stringers and limited masses of silicates in white limestone. Apparently as at Fausto Primero the ore solutions circulated through all the crevices and open spaces and produced the streaks of minerals by reactions with the limestone. Besides the iron ore, which is largely specularite, well crystallized garnets were produced in the calcite as is shown in Figs. 39 and 40. As the effects grow stronger and the conditions of the largest pit are reached, the silicates become more extensively developed. They may take on finely parallel growths with some surviving limestone and resemble coralline



FIG. 38.

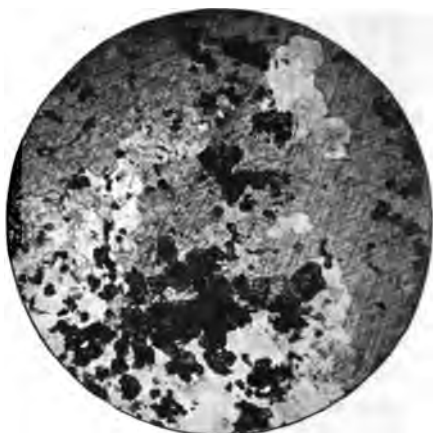


FIG. 39.

FIG. 38.—INTERGROWTHS OF MAGNETITE AND PERHAPS SPECULAR HEMATITE WITH FINELY PRISMATIC AMPHIBOLE AND CALCITE. ELVIRA MINE.

FIG. 39.—RELATIVELY FEW GARNETS (THE DARK MINERALS) WITH QUARTZ (THE CLEAR WHITE MINERALS) IN CALCITE. SIGUA MINES.

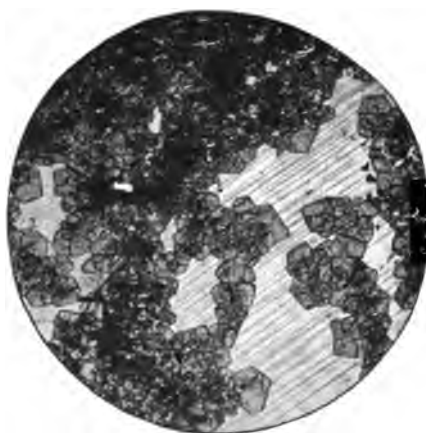


FIG. 40.



FIG. 41.

FIG. 40.—RHOMBIC DODECAHEDRONS OF GARNET IN CALCITE. SIGUA MINES.

FIG. 41.—PLATES OF SPECULAR HEMATITE IN CALCITE. SIGUA MINES.

structures, strongly reminiscent of *Eozoon Canadense*. Undoubtedly specimens like these, observed by Dr. J. P. Kimball, in the early days at Firmeza confirmed him in the idea of the replacement of coralline limestone. Sections of the contact silicates reveal all manner of interesting things. We may see yellow garnet, specularite plates, quartz, and diopside all in the same slide. Epidote appears in other slides. The specular hematite in the silicates shows a marked tendency to develop rosettes of radiating plates, Figs. 41 and 42, like roses, as one sometimes sees also in ilmenite. In the slides the plates ramify through calcite more often than through quartz. No reasonable explanation could be given of all these phenomena other than the standard one of contact effects. Some intrusive mass has fed into these limestones gases or



FIG. 42.

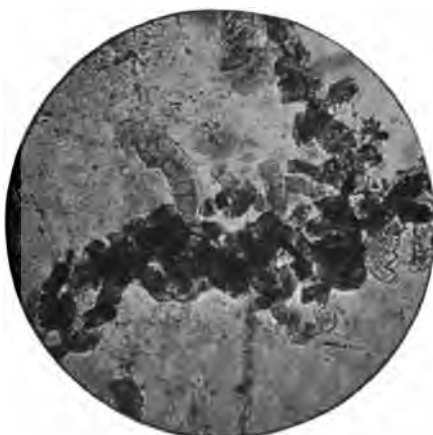


FIG. 43.

FIG. 42.—THIN, RADIATING PLATES OF SPECULAR HEMATITE IN CALCITE, BELOW; FINELY GRANULAR EPIDOTE IN CALCITE ABOVE. SIGUA MINES.

FIG. 43.—VERMICULAR GROWTHS OF SOME UNDETERMINED MINERAL IN QUARTZ. Actual field, 0.06 in. or 1.5 mm. Sigua Mines.

solutions, or both, which have brought silica, iron compounds, perhaps as chlorides or oxides, and have thereby reorganized the original limestones and produced the lime-iron silicates, the quartz, and the ore. From the mining standpoint the results have led, as in most of the actual contacts in southeastern Cuba, to irregular and seldom large bodies of ore in the midst of barren silicates so that a great quantity of waste must be broken and handled to secure a commercial output. Yet the residual blocks, left on the surface by weathering, originally gave the impression of great bodies underground, and led to the construction of the Sigua standard-gauge railroad to the sea, and to the building of a pier. Floods and the insurrection wrecked the installation after 20,000 to 30,000 tons had been shipped, and since then the Spanish-American company has obtained some 60,000 tons additional by means of a narrow-

gauge extension of the Berraco railroad. Even then 50 per cent. Fe and 12 per cent. SiO_2 limits were maintained. Operations have now been abandoned.

The ore appeared in various small open cuts down in the valley below as well as on the mountain side. There is obviously some complexity to the geology, because in the gulch below the main workings a stratified rock, believed to be an old volcanic tuff and breccia, was observed. In one exposure it had a strike N. 70 W. magnetic, dip 32 N., and again N. 80 W., 20 N. It indicated a marked change in formations from the limestone. In one of the lower pits the black shaly rock was observed, which was earlier mentioned as a probable fine tuff but now altered to a green chloritic mass. On a neighboring claim, the Alfeo, float ore was being picked up off the surface but no excavations were undertaken.

Between Sigua and the other mines much volcanic or at least porphyritic rock appears on the surface. It stretches along the narrow-gauge road for some distance to the south as a green andesite. In thin section the feldspar phenocrysts are plagioclase with chloritic remains of either hornblende or augites which are no longer recognizable. The ground-mass is finely microgranitic and contains some minute untwinned mineral which may be in whole or in part orthoclase, albite or quartz, probably the two former. A little magnetite is sprinkled through the ground-mass.

To the west of Sigua a purple porphyritic rock was gathered in several places. It proves to be a trachyte. Orthoclase is much more abundant than plagioclase among the phenocrysts, but the ground-mass is very finely crystalline, almost felt-like. Alteration is extreme; only limonite remains to suggest dark silicates. The rock was gathered in the general field work and we do not know whether the trachyte appears in dikes or surface flows.

Along the Sigua extension of the Berraco narrow-gauge railroad, and about two miles in a direct line, or three miles by rail southeast of Sigua, two shafts were sunk in former years in exploration of small veins believed to contain silver. Only the dumps and the open vertical holes now remain, but small specimens of barite apparently with tetrahedrite can be picked up. The wall rock is a dense greenstone, evidently derived from some rock like the diorites. No records are available of the assay value of the ore.

RÉSUMÉ

The iron ores are the subject of chief interest and in summing up, attention will be alone directed to them. They furnish a variety of deposits ranging from those of small size in streaks in limestone associated intimately and microscopically with quartz, garnet, and epidote;

through larger and more extensive developments of the same with the practical extinction of the limestone; to the extreme of ore developed in great tabular masses in diorite, but still associated with the same quartz, garnet, and epidote, both in the large and in the microscopic way. In the large bodies pyrite appears in increasing amount as depth is gained, but it has weathered out of the ore near the surface. The smaller deposits, obviously in the contact zones, have in no case been followed deeply enough to develop much if any pyrite and we can only infer its probable presence in depth. Pyrite is quite frequently associated with the magnetites and specular hematites customarily found in contact zones. As regards the magnetite and specularite of the obvious zones, there is little question that they have been yielded by iron-bearing waters or gases from the neighboring igneous rock. So many cases have been studied and so many observers have been led to the same conclusion that little attention is demanded by this type other than the mention of the method of formation. Experience has shown orebodies in contact zones to be irregular in shape and of no uniformity of disposition in the zones. They seem not to be very large in the Daiquiri district, although in the Iron Springs District of Utah and in some other localities in the southwest they are very large.¹⁸ In the larger and more important deposits in the Daiquiri district, such as those in Lola Hill and on Providencia Hill as developed especially in the Alfredo, and for the Old and New Norte, we cannot reasonably go far from a method of origin closely related to that of the contact zones. Their ore with its associated garnet, quartz, and epidote must be closely akin to that of the zones. The long and relatively narrow masses and the obvious location in belts of crushing and rock movement cannot be ignored. The observer is almost irresistibly led to the conclusion that emissions from some cooling, igneous mass beneath, bearing iron and silica, have come up along lines of faulting and crushing. We might justifiably wonder if where we now find the orebodies, slabs or blocks of included limestone chanced to be in the diorite and if they served as precipitants of the iron, so as at the same time to yield the associated garnets. The suggestion is by no means unreasonable for some of the smaller block-like masses, but as one appreciates the extent of the large bodies on Lola Hill, their great length and depth as compared with their width and their uniform vertical or nearly vertical position, the assumption of blocks of limestone is clearly so improbable as not to appeal to reason. The vertical and somewhat straggling line of ore which one sees at Providencia, Alfredo, and at the Old and New Norte could not be explained by it. No ex-

¹⁸ C. K. Leith and E. C. Harder. The Iron Ores of the Iron Springs District, Southern Utah, *Bulletin* 338, U. S. Geological Survey, p. 73 (1908). A total of 40,000,000 tons is estimated, the larger part in contact deposits, of which one reaches 15,000,000 tons.

planation will fit except one along the lines of vein formation with replacement of the diorite. Where the result was largely magnetite, but so mixed with silicates as to be too low in grade for ore, "piedra azul" resulted. Where the wall rocks yielded more completely to the iron solutions or vapors commercial ore resulted. The diorite was replaced with magnetite and some specular hematite, intermingled with garnet, quartz, and epidote. The lime for the admixed garnet and epidote probably come from the lime in the plagioclase or in the bisilicates of the diorite. But in the case of the streak of garnet rock in the San Antonio with admixed calcite but no appreciable iron ore, an included slab or series of slabs of limestone is not unreasonable. As stated above, veins of magnetite are known in the diorite porphyry or andesite which has developed the large contact bodies at Iron Springs Utah. E. P. Jennings has considered them iron dikes, but Leith and Harder regard them as due to emissions from the igneous mass, coming up through cracks in its outer cooled and solidified shell. In some such way as this the Cuban ores have probably been formed.

As was inferred by F. F. Chisholm there seems no reason why the ores should be necessarily limited to relatively shallow depths, but with depth, unoxidized pyrite is quite certain to appear and to contribute relatively high sulphur. Roasting is already practiced at Firmeza.

The larger deposits thus lie along structural lines. The resistance of the ore to weathering processes has left it as the backbones and at the summits of hills whose slopes were mantled with blocks of float. Meantime the original wall rocks have weathered away and have in part been removed by erosion.

The Daiquiri ores are remarkably similar in their geological relations to a deposit near Hong Kong, China, recently described before the Institute by C. M. Weld,¹⁴ and to others in the province of Bulacan in the Philippines as set forth by Dalburg and Pratt,¹⁵ and associated petrographic studies by F. T. Eddingfield.¹⁶ In each case aggregates of preponderating magnetite with specular hematite have resulted by contact metamorphism and replacement. The geological sections in Bulacan might almost be substituted for those in southeastern Cuba.

¹⁴ C. M. Weld. *Bulletin* No. 86, Jan., 1914, p. 177.

¹⁵ F. A. Dalburg and Wallace E. Pratt. The Iron Ores of Bulacan Province, P. I., *Philippine Journal of Science*, vol. ix, No. 3, p. 201 (June, 1914).

¹⁶ F. T. Eddingfield. Microscopic Study of the Bulacan Iron Ores, *Idem*, p. 263.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Tramming and Hoisting at Copper Queen Mine

BY GERALD F. G. SHERMAN,* BISBEE, ARIZ.

(San Francisco Meeting, September, 1915)

THE ore deposits of the Warren district, in which the mines of the Copper Queen Consolidated Mining Co. are situated, have been described in a number of technical publications, and will not be discussed here in detail. Certain of their characteristics, however, control methods of underground transportation and hoisting of ore.

In the Copper Queen mine, the majority of the ore has occurred in a zone encircling the west boundary of the porphyry intrusion of Sacramento Hill. It has a width varying from 800 to 1,200 ft., and a thickness of about 400 ft. It reaches the surface in the older part of the mine to the northwest, but dips to the southeast, where it is reached at 1,400 ft. below the Czar collar, in its farthest extension at present developed on Copper Queen ground. There is one major extension from the northwest end of the zone toward the west along the Czar fault, and others of minor importance.

Individual orebodies are scattered through the zone in an eccentric manner, only matched by their own irregularities of form and size. Their most general characteristics are the softness of the ore and their great horizontal rather than vertical extent. It has been estimated that the average vertical thickness of ore in the Czar and Lowell divisions is between 30 and 35 ft. It is calculated by assuming it to be uniformly distributed over its horizontally projected area.

In this zone, and for some distance above it, the ground has been subjected to intense alteration and intense but irregular oxidation. It has resulted in an enormous quantity of earthy or clayey material, which may be either ore or waste, which when wet is both heavy and tenacious. Below alteration, the ground is fairly hard, and the limestones contain primary ores differing from those heretofore considered typical of the camp, and which have not yet been thoroughly exploited.

The mine production has been drawn from many orebodies spread over a great area. A diagram map (Fig. 1), of haulage drifts and loading stations, illustrates the situation in 1914.

The effort to restrict the area of work and intensify production has not been particularly successful, for several reasons:

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First: The mine has been in continuous operation for more than 30 years. During the earlier period, the costs of mining and smelting were high, and ores which are now profitable could not be worked. Exploration was not carried far enough in advance of mining to explore orebodies as a whole, and they were attacked piece-meal. Much ore was, in consequence, left in and about the old stopes. When new ground was opened to supply the demand for greater production, the old areas were not yet exhausted, their output was merely reduced. In fact, more thorough exploration and the gradually decreasing cost of mining, etc., has actually increased the production and reserves of the Czar division, the oldest part of the mine.

Second: As all ore is smelted direct, a balanced production must be made to maintain a self-fluxing mixture. Since the oldest divisions, the Czar and Holbrook, have contained highly refractory ores, their output has been limited to what the smelter could handle, and their reserves have not been exhausted as rapidly as the economies of mining alone would suggest.

Third: While many of the orebodies are large when taken as a whole, they are small in section, complicated in form and mingled with waste. They cannot be mined rapidly.

The producing area has therefore increased rather than diminished.

All the ore is not sticky or hard to handle, but some is extremely so. To one not acquainted with it, it is hard to appreciate how seriously this quality interferes with every operation of handling, and forces complications into a general plan to insure certainty of action. There is not such a large proportion of sticky ores at present as in the past, except on the 400 level, but a successful method must be prepared to take care of it wherever it appears.

To summarize briefly: The three characteristics of the deposits which affect haulage are scattered operations, heavy ground in and near the ore, and clay ores.

Old Haulage System Unsatisfactory

Prior to the introduction of the system now in use, the ore was trammed by hand to one of several shafts, where it was hoisted on cages and loaded into railroad cars for shipment to the company's smelter at Douglas. In most cases, it was dumped directly into the cars instead of loading from bins, since it was slow and hard work to get it out of storage once it was put in. An attempt was made to mix the ore when loading into the railroad cars, but without great success. Each tenth mine car was loaded into a separate car as a sample for the smelter, but the returns were not beyond question.

An unusual number of shafts were required, since active production covered a wide area. Distances to the shafts were excessive for hand

tramming, and it was impossible to maintain such great lengths of drifts and track in proper condition, although heavy repairs were continuous.

The defects of the system became more and more apparent as the mine grew, and tentative plans had been discussed from time to time, but the disadvantages of any new method were more apparent than the advantages, and nothing was done.

Attention was again and more forcibly drawn to the subject when the sudden loss of the Holbrook shaft in July, 1906, diverted more ore to the Czar and Spray than they could well handle. Tramming costs were raised by the increase in the distance trammed, and by congestion at junction points or at the stations. This resulted in heavy expense, which lasted six months until the new 559-ft. shaft could be finished and equipped.

Plans were begun on a general system for tramming and hoisting, which were later extended to include a power plant, and thus began a centralization of the surface plant and work, which has only recently been completed by the construction of a central timber-framing mill and a drill- and tool-sharpening shop.

Formerly, ore was hoisted through five shafts. Beginning with the oldest and shallowest, and passing to the southeast, they were as follows:

Shaft	Depth	Production of Dry Ore per Month, 1906
Czar.....	417 ft. to 400 level	7,814
Holbrook.....	559 ft. to 500 level	17,263 (1st 6 mo. only)
Spray.....	1,042 ft. to 900 level	18,530
Gardner.....	1,032 ft. to 1,000 level	7,560
Lowell.....	1,215 ft. to 1,200 level	4,327
Sacramento.....	928 ft. to 1,000 level	
Total for year.....		564,154

The Holbrook has since been sunk to the 600 level and the Lowell to the 1,600.

Of these, the Holbrook, Spray, and Lowell shafts were either in the ore zone or so near stoped areas that there was continual movement in them, and some little danger of a repetition of the Holbrook collapse.

Plumbings were made regularly to ascertain the extent and rapidity of movement in the new Holbrook shaft. During the first two years, the collar moved to the northeast at the rate of an inch per month. It is now 6 ft. 2½ in. N. 45° E. of the stationary part below the third level.

Although the Spray shaft is in rather solid limestone, a crack developed, which passed through the shaft and split the engine foundation.

It may be noted in passing, that the engine foundations at these shafts now consist of two massive blocks of concrete, reinforced with old rails,

each heavy enough in itself to hold the engine securely, and resist the rope pull. The ground beneath was levelled off by a layer of concrete, but the surface was separated from the foundation block, so that there could be no bond which would tear the concrete block apart in case of future movement. The fissure at the Spray shaft has since widened, but with no more effect than to tilt the foundation backward, engine and all. It runs smoothly on a slope of $\frac{3}{8}$ in. in 1 ft.

The Czar, although in rather heavy ground, was stable, and the Gardner is outside the ore zone and will probably remain true.

Each shaft was an independent unit, with its own boiler plant, etc. The equipment was miscellaneous, having been added to from time to time, and some of it was very old. The operation of the several plants was inefficient, from the number and small size of the units. The use of power was unimportant aside from that for hoisting.

New Plan Included Electric Haulage and Central Hoisting

It was proposed to equip a shaft, located in some central and safe position, for economical hoisting, where the ore might be mixed and sampled. The ore was to be drawn to the shaft by electric locomotives.

The advantages to be expected may be stated as follows:

1. The hand tram would be cut down to an average of 300 ft. or less.
2. Repairs would be reduced by diminishing the number and length of tramming drifts in bad ground. Access to stopes may be by circuitous routes to avoid heavy ground, and need not be so well maintained as is necessary when large quantities of ore must be drawn through them, the main-haulage drifts being placed beneath the ore in solid ground.
3. Fewer trammers would be required, and they could be distributed to avoid interference. There was often great congestion near the shafts, even when mining less ore than in recent years.
4. Hoisting would be cheapened by the use of skips, a more efficient engine, and a better method of transporting the ore to the railroad cars.
5. The unstable shafts would not be so essential to continuous production.
6. Shaft and cage repairs would be diminished, particularly in the moving shafts, by slowing down the hoisting speed, as could be done if only men, timber, and waste were hoisted.
7. Miscellaneous work about the shafts and auxiliary equipment might be cut down.
8. Power distribution from a central plant would be simplified and the investment for hoisting machinery reduced.
9. The ore could be thoroughly mixed and accurately sampled before shipment.

There were also certain disadvantages, aside from the cost of the improvements:

1. Haulage drifts must be driven and maintained without abandoning a corresponding length of hand-tramming drifts.

2. The ore would be transported underground a greater distance than before. A part of the saving in the length of the hand tram would be lost in the added operation of power haulage.

3. The old shafts must still be maintained for men, timbers, and waste rock, while the main shaft is added to those in use. (The new State law restricting the time of underground labor to 8 hr. from collar to collar, has made it important to have shafts close to the stopes and active workings.)

The advantages were believed to outweigh the disadvantages, and the construction was authorized.

Electric Haulage on Alternate Levels Only

The cost of equipping each level for electric haulage was prohibitive. Instead, each alternate level was chosen, beginning with the 400, making seven in all to the 1,600. By collecting the ore from 15 producing levels to seven haulage levels, the traffic density is increased, although still far below that necessary for really cheap transportation.

By referring again to the haulage map, it will be seen that the 400 level draws ore from the whole of the shallow western end of the mine. It begins in the solid ground under the stopes, passes out to one side of the ore zone and then to the hoisting shaft. It was possible to select old drifts in stable ground for most of the main line. The only new drift of any length was from the Gardner to the Sacramento. Ore is dropped from the 100, 200 and 300 to this level, through transfer chutes, under or near the stopes. In some cases, the haulage cars are collected by hand through short branch drifts to the stopes on the haulage level; in others, when the quantity warrants it, the trolley wire is carried to the stope chutes direct. Of the whole 5,600 ft. of main-line trolley drift on the 400, not more than 1,500 ft. was timbered, and of this only 450 ft. is particularly heavy.

In a similar manner, the 600 level was driven through from the Sacramento to the Gardner, old drifts were used to the Spray shaft, which were extended by new drifts, to reach the ore as it dipped below the 400 level. It underlaps the 400 haulage for some distance. Although this drift passed through the ore zone between the Gardner and Spray, some of the old drifts were in good ground, and the proportion of heavy ground to sound is not much greater than on the 400.

The same plan was carried out for the other levels, each one collecting ore from an area deeper, and farther to the south and east, than that served by the level above. Not very many new drifts were driven particularly for this purpose, and the percentage of main drifts which require repairs is small.

A trolley line on the 200 level to the Uncle Sam shaft was not contemplated in the beginning, but was added afterward. The ore from that shaft and the Southwest country is hauled to a transfer chute near the Holbrook shaft, where it is dropped to the 400. Extending the 400 level to the Uncle Sam to avoid re-handling may be justified at some later date.

To begin with, 7,000 ft. of new drift was driven especially for power haulage, but other drifts have been driven since, and others equipped for the purpose, until at this date, 10.9 miles of track and trolley line are in use or are standing ready. The connecting drifts on the 1,200 and 1,400 levels were warranted as prospects, and many others now used for haulage were driven to find or develop ore. Probably not more than one-quarter of the whole should be charged to the haulage system.

Track is 20-in. Gauge

Twenty-five pound rails on 20-in. gauge are used for the track, with 4 by 6 by 42 in. ties. They have been heavy enough except in a few cases.

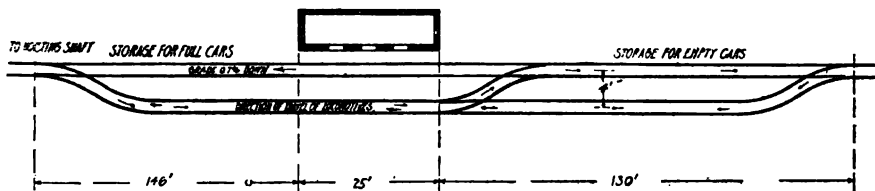


FIG. 2.—TRACK LAYOUT AT LOADING STATION.

The maximum concentrated load is on the 400 level. Locomotives are there used weighing 14,000 lb. The weight of a loaded car rarely exceeds 6,000 lb., although it may be more when loaded with massive sulphide ore. Wet ground on this level caused very heavy track repairs; even when slag was used for ballast, it was churned out from under the ties. To improve this condition, 4,200 ft. of second-hand 40-lb. rails were laid to replace lighter rails. The stiffer rails distributed the load better, and repairs were much reduced.

As a rule, the train crew does not load the ore. The motorman brings empties to the stope or transfer chute, and takes out the full cars standing there, or returns when the others are filled.

In the larger stations, sufficient double track is laid for tail room to hold 20 cars. A standard track layout is shown in Fig. 2. Only one switch, at A, is thrown by hand; spring switches direct the locomotives without further attention. At unimportant chutes, less elaborate sidings for car storage are used, or the trains may be made up in the stope drift or on the main line.

At the Sacramento shaft, the tracks are looped to avoid switching. Standard switches on curves of 40 to 80 ft. radius are used. Curves of 25-ft. radius are permitted only in exceptional cases.

Direct Current at 250 Volts Used Underground

Direct current is carried into the mine at 250 volts, through the Sacramento, Gardner, and Holbrook shafts. The feeder cables are cambric insulated, lead covered, and wire armored. The return cables have weather-proof insulation only. The voltage on the west end has been low, for lack of capacity in the power cables. A transformer set has been purchased to be set up at the Holbrook shaft to serve as a booster, but it is not yet in service.

The current underground is carried on 00 B & S American standard grooved trolley wire. The rails are bounded with No. 12064 Type E O. B. rail bonds, length 32 in. center to center, capacity B & S. No. 0, diameter of compression terminals $\frac{5}{8}$ in.

Trolley wires are now required by law to be 7 ft. above the track. They are hung from the timbers in timbered drifts, or from 4 by 6 in. cross bars held by vertical bolts wedged into drill holes, when the ground is hard. The old lines were sometimes hung lower, and in such cases were protected by 1 by 6 in. battens placed parallel with the wire, 8 in. apart and projecting 2 in. below the wire. On straight track, the wire is supported every 20 to 25 ft., as very little sag is permissible. Occasionally, the wires must be protected from acid water dripping from the roof. There is also more or less trouble with bonds in wet drifts.

There have been five fatal accidents in which the trolley current has been involved:

No. 1, September, 1908. Pipe-fitter at work on air line above wire. Received shock and fell to track 7 or 8 ft. below. Uncertain whether shock or fall was fatal.

No. 2, December, 1908. Miner repairing drift stood on a truck and fell off. Neck was broken. Thought that shock from trolley wire caused the fall.

No. 3, November, 1909. Loader carried chute bar when switching cars and struck wire beyond protection.

No. 4, May, 1912. Miner climbed over couplings of standing train. Tilted steel on shoulder and touched trolley wire.

No. 5, July, 1914. Miner repairing drift. Ground fell, carrying down trolley wire and pinned him down with wire against his back. May have been killed by the fall. Current was left on trolley wire to light the drift, contrary to rules.

The first two cases were doubtful, as the shocks may not have been sufficiently severe in themselves to cause death. The danger of a current

of 250 volts was not realized at once, as it is the lowest in common use, and other mines have not found it so. Other men also in this mine have received shocks without injury. It was supposed to be due to unusual susceptibility on the part of the individuals. It now appears most probable that when shoes are wet, as often happens, with acidulated water, and there is moisture on the hands or body, an unusually good contact may be made, with fatal results.

When this condition was recognized, special precautions were taken to prevent accidental contacts. The wires are high enough to avoid being touched ordinarily. Any reasonable increase in height would not entirely prevent contact when carrying tools, etc., on the shoulder, which is also contrary to rules.

The greatest danger is in repairing drifts or loading ore. No repairs are now permitted in the vicinity of live wires, and the system has been divided to facilitate cutting out the current in sections under repairs, without interfering with ore haulage. At loading stations, the wires were at first protected by boards on each side of the wire. The current is now switched off the wire on each side of the chutes for such a distance that the loader cannot receive a shock while loading, or when switching cars, even in case he should carry a chute bar with him when doing so. When the loading station is on a main line, the current is carried past on an insulated cable, and is only turned on the trolley wire when the train passes and after warning the loader. Current in the wire is indicated by a light.

There are a number of loading stations on each level, and as it takes as long or longer to load, than to make the trip to the shaft, dump, and return, the locomotive usually draws from more than one station. On the 400 level, where the ore is most difficult to handle, and there are many loading stations, they are connected by wire to a signal box containing an annunciator similar to that used in hotels. The loader can thus signal to the motorman when his train is nearly full, and save a trip to a station where the train is not loaded.

Red lights are used in a crude hand-operated block system when there is danger of interference with timber or waste handling, or when there is a second locomotive at work on the same level.

Colored lights are also used to indicate the position in which switches are set, or the location of a place of refuge from passing trains.

Factors Affecting the Choice of Cars

So many of the old drifts were to be used when the system was planned, that it seemed impossible to use large cars.

The first cars bought held 21 cu. ft. They had gable bottoms, with hinged sides, and were equipped with M. C. B. Midget couplers, spring-

draft rigging and roller-bearing axles. The latches were to be sprung for dumping at the shaft bins, by a block between the rails.

This type of car was not satisfactory. A large proportion of the ore is too sticky to slide out of the cars, even when the doors are held open. It was customary on the 400 for two men to rock them from side to side on the rails, in order to clear them of ore. The frictional resistance to dumping increases as the size diminishes.

Another disadvantage was their tendency to leak fine ore, particularly if wet.

After using them for some months, a few rocker dump cars were designed and built, of about the same capacity. Spring-draft rigging, automatic couplers and roller bearings were used, as they had been found advantageous. The rollers in the first bearings were solid. For the new cars, an axle was designed to use Hyatt rollers, with special care to make them dirt-proof. They dumped freely, and were otherwise satisfactory.

Since it was necessary to secure a complete new equipment of cars, the question of size was re-opened, and a car of 33 cu. ft. capacity was adopted.

The cost per cubic-foot capacity decreases as cars increase in size. They should cost less for repairs also, in proportion to the ore carried, as the details of latches, etc., are heavier and stronger. Larger net loads can be carried in large cars, as the weight does not increase in proportion to the capacity. The cost of loading and particularly unloading is generally less. For these reasons, the largest car possible should be used.

The size of the cars in this case was limited by the cost of enlarging the drifts, and by the weight which can be conveniently handled by one man.

A large sum of money was spent in widening drifts for the 33-ft. cars. A car enough larger to be of appreciable benefit would have incurred prohibitive expense in widening the $4\frac{1}{2}$ miles of drift then in use, and an added expense for all haulage drifts to be driven in the future.

At important loading stations, the track is laid on a grade to favor the loader in assembling cars. In many cases, the quantity produced will not warrant special sidings, and the cars must be loaded and pushed by hand to the point of assemblage. The 33-cu. ft. car weighs 1,700 lb. Loaded with dry, oxidized ore it weighs about 4,250 lb. When loaded with sulphide, it may weigh as much as 7,500 lb. This car with roller-bearing axles can be readily moved by one man at all transfer chutes, and through branch drifts in most cases, unless the grades are unfavorable. A larger car would quite frequently need two men. The use of cars larger than one man can move would increase the expense of switching by hand, until a car twice as large is used, while that of loading would

be practically doubled, since there is only room for one man to work conveniently at a chute.

The rolling-dump type was selected for its ease of dumping sticky ore, and simplicity of construction. Tipples were not seriously considered, as seven would have been required.

The advantages of such refinements as spring-draft rigging and automatic couplers may be open to argument, but a few fingers have probably been saved by the latter.

The roller-bearing axles have been a most unqualified success. They are made in two sizes, and are of as great advantage on small cars as large. The axles on the motor cars are filled with grease once a month; on the hand or mule cars once in two months. It is probable that this time is shorter than necessary, and experiments are being made to determine how long the charge will last. There has been an economy in lubricants accompanied by a very low depreciation charge. A few of the first Hyatt bearing axles, in use since January, 1909, have begun to fail. The cage goes first, which allows the rollers to become displaced and injured. These axles are $1\frac{3}{4}$ in. in diameter and are used on 22-cu. ft. cars. They are rather light for the service. The larger cars have $2\frac{3}{4}$ -in. axles, which appear to be amply large for the load, and a longer life is expected.

The friction load is very light. Unfortunately, the average mileage per car cannot be known.

TABLE I.—*Wear of Car Wheels, Axles and Roller Bearings*

Size of Car, Cubic Feet	Wheels				Roller Bearings			Axles	
	Date Put in Service	Material	Original Diameter, Inches	Wear in Diameter, Inches	Make	Original Diameter, Inches	Wear in Diameter, Inches	Original Diameter, Inches	Wear in Diameter, Inches
22	Jan., 1909	Manganese steel.	14	0.44	Hyatt	$\frac{9}{16}$	0.053	$1\frac{3}{4}$	0.013
33	June, 1910	Cast iron	14	0.41	Hyatt	$1\frac{5}{16}$	0.0075	$2\frac{3}{4}$	0.026
33	June, 1910	Cast steel	14	0.375	Hyatt	$2\frac{7}{32}$	0.008	$2\frac{3}{4}$	0.04
33	June, 1910	Cast iron	14	0.34	Solid	$2\frac{7}{32}$	0.023	$2\frac{3}{4}$	0.062

The roller bearings on which the above tests were made were manufactured either five or six years ago. The Hyatt Roller Bearing Co. advises that the rollers are now made of chrome-nickel alloy steel, which has a very much higher elastic limit than the steel used in bearings formerly.

The limits of accuracy in finishing are now approximately 0.002 in. instead of 0.005 in. It is very probable that better results would be obtained with the rollers as they are now made.

Several Types of Cars Used.

Several types of cars have been used, and a summary of their characteristics may be of interest.

The original car had a capacity of 12 cu. ft., and was of an ordinary type, dumping forward and also, by swivelling, to the side. It is not worth serious consideration, as it was both light and weak. The older shafts were small, and a car of greater capacity of this construction could not be put on the cages.

Two cars of 16- and 20-ft. capacity each (Fig. 3), dumping only to the side, were recently designed after a car developed by the Old Dominion Copper Mining & Smelting Co. The 16-ft. car will go on the cages in the smaller shafts. The 20-ft. car can only be used in the Gardner,

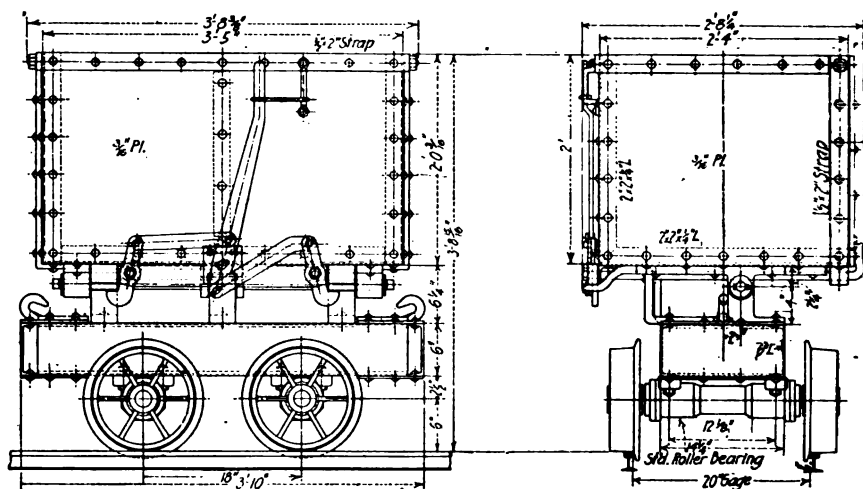


FIG. 3.—16-CU. FT. SQUARE SIDE-DUMP CAR.

Sacramento, Spray, and the newer prospect shafts. These cars have great capacity for their over-all dimensions. They are of heavy construction. They can be dumped without uncoupling when used in trains. This is an important advantage, as they are frequently hauled in short trains by mules or motor to transfer chutes, or in carrying waste from development work. It is a disadvantage that it can only dump to one side, and care must be taken to see that it comes to the dumping place faced the proper way. It is proposed to replace the 12-ft. cars with these as fast as they wear out. It is unfortunate that two sizes need to be used, but it is believed that the added capacity of the large car makes it worth while.

Rolling-dump cars of extremely simple construction were designed for use at the North Star Mines, in Grass Valley, Cal., and were introduced

at the Copper Queen for surface work. Afterward, they were taken underground, and have proved satisfactory for hand and mule tramming. They are cars reduced to their lowest terms, consisting as they do of a pair of axles with wheels, two beams of "I" or channel section tied together at the ends by two cast-steel supports for the body, which also form bumpers and attachments for connecting in trains. In small sizes, they may be locked by a pin through trunnion and support, if necessary. When properly balanced, they dump with great ease, and do well with sticky ore. As the body forms a tight box, ore is not spilled, even if it is a thin mud. As they dump more easily when well filled, the tendency is to heap them up rather than load them to only a part of their capacity.

They are bulky for their capacity, however, and while this is not important when used exclusively on levels, it is an objection for cage work. They are also high and not so convenient to shovel into. In order to prevent a multiplicity of types, it is probable that the square-bodied side-dump car will eventually supersede these also, as they can only pass through the newest shafts.

The 33-ft. car of this type, which is used for haulage, is too large for general hand or mule tramming, and has most of the advantages and disadvantages of a small car of the same type. It dumps muddy ore more readily than any car which has been tried, but if the ore does not all slide out together, what remains is in the bottom and it is hard to hold in position to dump. The point of support is placed below the center of gravity in order to dump easily, and a lock on the bottom prevents its over-turning in transit. A larger-sized rolling-dump car is not to be recommended. The force necessary to revolve a weight of 2 to 3 tons may be excessive.

In larger sizes also, the objections to the saddle-back car disappear. They dump ore well when they hold 75 to 100 cu. ft. The doors are the only moving parts, the ore is merely permitted to fall out, and, if it does, nothing better can be desired.

Three to 7-Ton Electric Locomotives in Use

There was difficulty at first in securing standard locomotives small enough to go through the drifts; 3-ton Goodman locomotives were selected, their type No. 1600-K. They were small and compact, and were driven by a single motor of 20 to 25 h.p. on one-hour rating. Since, under ordinary conditions, the train crew does no loading, the locomotives were in continuous motion. It gave the motors no time to cool off, armatures burned out, and electric repairs in general were very heavy. About this time, it was decided to use larger cars, the drifts were widened and larger locomotives could be used. Two 6-ton locomotives (6 N-0-2) were then purchased. They were driven by two 8-A Goodman motors of 22 h.p. each on one-hour rating, and have been satisfactory.

The haul on the 400 level is long, with adverse grades for a third of the distance. A still larger locomotive was thought desirable for this level, and two 7-ton machines were designed and built in the mine shop. They are extremely small for their weight and power. The frame is of cast steel. The two Westinghouse No. 79 mining motors, of 30 h.p. each on one-hour rating, were taken second-hand from stock. In the design, everything was sacrificed to compactness and power, and it is not to be recommended for general purposes. The unusually rugged electric equipment has fulfilled every expectation in the elimination of delays and repairs. The tire expense is rather heavy, since the clearance below gears is so small, the tires can only be worn down 1 in. in diameter before replacement.

The small locomotives replaced by the heavy machines have been used to good advantage on light work, either on levels where the ore production is not heavy, or in hauling waste or timber. They were well constructed, serviceable machines when used with the load for which they were designed. Since then, repairs have been general and not confined to any particular feature. Gears and tires make up most of the operating cost. The gears are cut out of special steel blanks on a milling machine at the mine shops. The tires are of forged steel shrunk on a cast-iron center. The treads are re-faced when necessary, and are worn down to a thickness of $\frac{1}{4}$ -in. at times. Manganese-steel tires were tried, but were found to wear very rapidly.

Odometers have only recently been attached, accurate mileage figures therefore cannot be given.

In narrow drifts, it is sometimes impossible to turn the trolley pole, and many were broken in backing up. A trolley pole with a knuckle in the center, which is easier to turn and which can be turned anywhere, has reduced the breakage.

In selecting a locomotive, it should be seen that the motor equipment is large enough for the work to be done; it will save endless trouble, delays and expense.

Power-Rating of Locomotives per Ton on Wheels

Size Locomotive, Tons	One-hour Rating, Horsepower	Horsepower per Ton
3	20	6.6 to 8.3
6	44	7.3
7	60	8.6

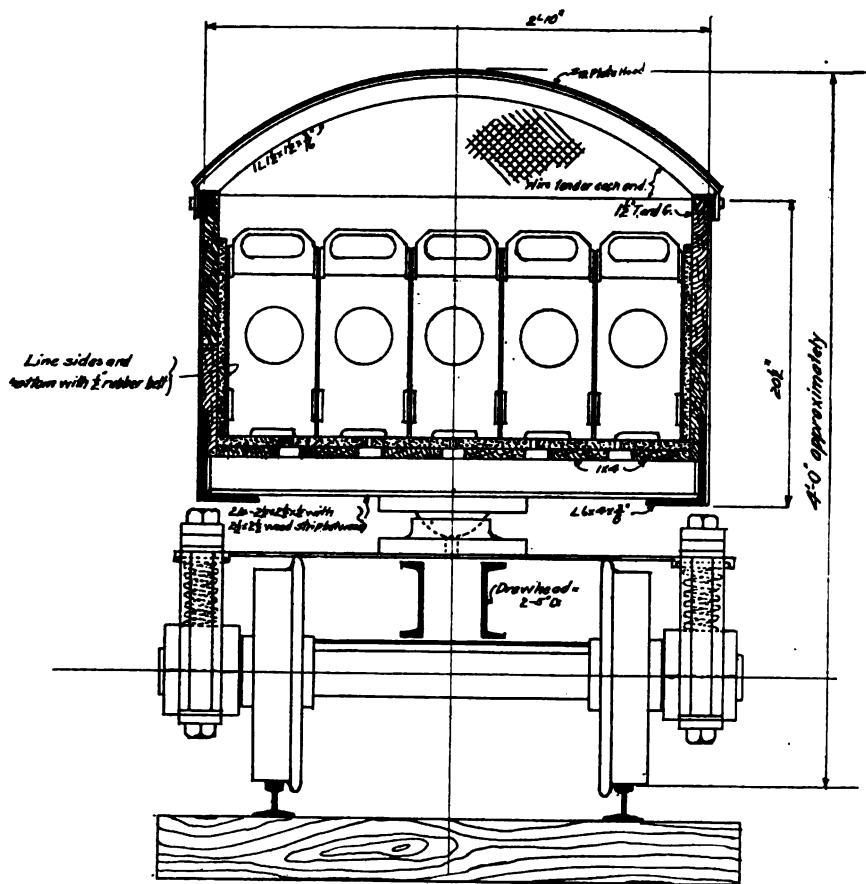
Storage-Battery Locomotive Tried

One of the 3-ton locomotives was used for a time in 1910 and 1911 with an Edison storage battery. The results of the running test were presented in a paper by Charles Legrand.¹ The battery was composed of 150 A-6 cells carried in two separate trailing cars.

¹ Electric Traction in Mines, *Trans.*, xlviii, 295 (1914).

Moisture from the air accumulated on the cells, to which was added a small proportion of electrolyte as a result of spillage or gassing while being charged, and a number of cells were lost by short circuiting.

In 14 months, 21 cells were burned out. The whole battery was then returned to the factory, and replaced by another with different crating to provide better insulation. A new battery car provided with springs



Cross-Section of second Car

FIG. 4.—UNDERGROUND MOTOR STORAGE-BATTERY CAR.

(Fig. 4) was built large enough to contain them all. It weighed 2,400 lb., and the battery 3,000 lb. The cells were occasionally wiped dry, painted and coated with vaseline. The nickel cell walls of only two cells were burned through in the 7½ months of its service. The cells at the ends of the cars were those usually lost.

If the batteries could have been kept perfectly dry, or if the voltage

had been lower, there should have been little or no difficulty from short circuiting.

When idle and fully charged, the voltage was 213, after travelling about 8 miles, one-half under load, the voltage dropped to about 200. When the voltage dropped still lower, the speed of the locomotive decreased and the work went slowly.

Ore was drawn during the day and the battery was charged at night, by current from a 20-h.p. motor-generator set taking power from the alternating-current circuit. It was automatically cut out when charging was complete.

The production from the area served by this locomotive gradually increased until its capacity was exceeded. It was then thought better to lay a wire than to get another battery, and the use of the storage battery was discontinued. The cost of the locomotive was approximately \$1,300, that of the battery about \$3,000.

The operating costs were good in spite of light rails and sharp curves. No information on normal depreciation can be given.

Favorable conditions for motor and battery are opposed. In order to protect the battery, the motor must not be too large. And yet it has been found that motor repairs are reduced by making them powerful enough to slip the wheels of the locomotive readily. The added weight of the battery gives them good traction, and there is always a temptation to pile on the load. By incorporating the battery with the locomotive, they may be better suited to each other and the work to be done.

The capacity is not very great. The cost of battery, charging station, etc. is to be balanced against that of trolley wire and rail bonding. The charge for power is so small that the greater consumption of power-house current is of no particular consequence.

The battery is now set up in the power house to excite the fields of the alternating-current generators when starting. They replaced a small steam-generator set, and have been very useful.

Several Designs for Stope and Transfer Chutes

When the stopes are on a haulage level, only the regular stope chutes are used, and the cars are brought to them by the locomotives, if the tonnage is of sufficient importance, or by hand from some nearby point where they may be re-assembled into a train to be hauled to the shaft. If the ore comes from the level above, it is dropped through a transfer chute to the haulage level. This may be an old stope chute or it may be a special raise into which the ore from a considerable area is collected.

For the larger bodies, a chute is usually provided close to the stopes, even if a special drift must be brought in from the main line. In a top

slice, the main chute is brought up to the top floor and the ore is trammed from the working faces to the haulage chute direct. In the Dividend slice, the ore being too soft for raises to be maintained through it, they are placed in the country rock outside, and the ore will be trammed a longer distance on the working floor of the slice, instead of to a central raise within the ore.

In the Howell slice, the ore mined above the 600 is dropped to the 1,000, as it is economically impossible to maintain a trolley drift through the ore zone on the 800. In this case, the ore is trammed again on the 900, as there is not sufficient to justify a special raise clear through.

Table II.—Loading Cost for Levels in 1914

Level	Cents per Ton
400.....	6.4
600.....	2.4
800.....	3.4
1,000.....	3.5
1,200.....	1.8
1,400.....	2.8
1,600.....	3.2

The 400-level costs for loading are highest, since the greatest proportion of clay ore originates in the area tributary to that level. The ore in the Dividend stopes is a very tenacious clay, and a cost of 10c. per ton for loading has not been uncommon nor excessive. The difficulties of loading have directed particular attention to the development of a better chute than that formerly employed.

A rectangular vertical chute of small section, with a bottom sloping down to a door in the front, is the worst possible for sticky ores. The small section increases side friction and favors arching. The weight of the column of ore packs it tightly into the wedge-shaped section at the bottom, particularly when it falls as a mass after hanging up. Only a part of its weight acts as a force to push it horizontally through the door. By shortening and widening the column of ore, the weight per square foot on the bottom layer is lessened, and there is less possibility of arching. By putting the door underneath instead of at the side, the full weight of the ore acts to force it through. These principles were used in developing a chute which has been found to be of great advantage. The doors have been put underneath the bin in only one case, as it is seldom necessary.

From the sketch of a standard transfer chute (Fig. 5), it will be seen that the storage is all at the bottom. The ore falls through the chimney and strikes a baffle, its downward velocity is checked and it falls lightly on the heap below. The pile of ore is not allowed to extend up into the raise, which serves only as a passage way. The storage space is long enough to let it spread into a loose pile, there is little tendency to

pack, and it cannot readily arch. Below, there are several doors. If, when loading, the ore hangs up over one door, the car is moved to the next, which will break the arch, or if that door does not, to the third. The bin itself is usually placed in solid ground and requires no timber except in the bottom. The narrow chute above is only lined in heavy ground, and then usually with concrete. Its small diameter, from 3 to $3\frac{1}{2}$ ft., brings this expense within reason.

Seven chutes have been lined with concrete. One which has been in use for 20 months, needs re-lining, due to wear by sharp ore. This

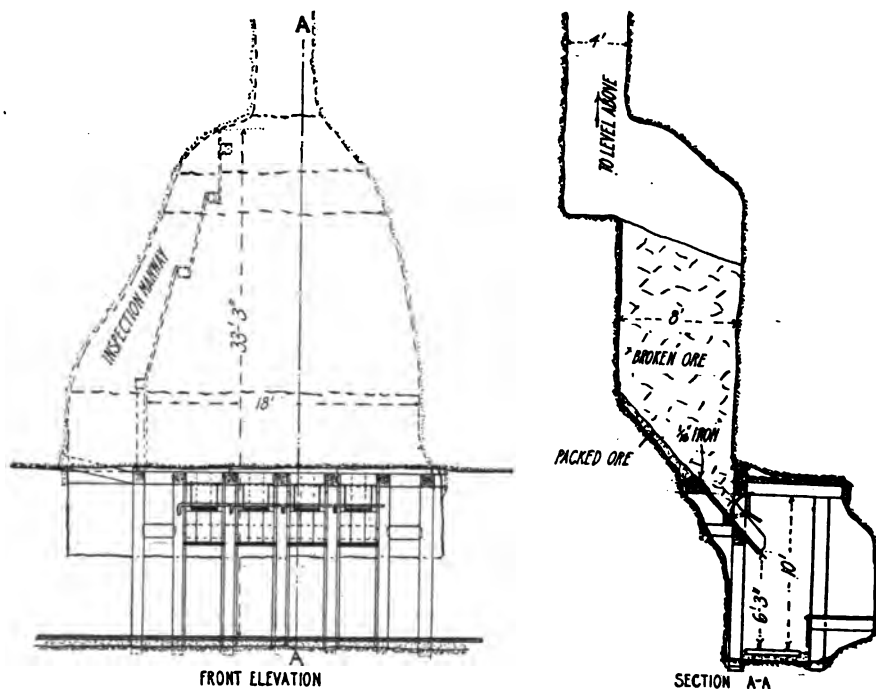


FIG. 5.—STANDARD HAULAGE CHUTE. 90 TONS CAPACITY.

chute would have required heavy repairs if timbered, both to provide against outside pressure and to replace the lining. In re-lining, a thicker wall of concrete will be used. Other chutes are only slightly worn.

An elaborate storage bin, shown in Fig. 6, was built on the 400 level to serve a large aluminous orebody which has been developed, but not yet mined. It was described by Joseph P. Hodgson, Mine Superintendent of the Copper Queen mines, in a paper read at the Salt Lake meeting of the Institute.² A large tonnage of ore is tributary to it, which is so sticky that extreme measures were considered necessary.

² Mining Methods at the Copper Queen Mines, *Trans.*, xlix, 316 (1914).

This is the case in which the doors are put underneath the center of the pile of ore. There is an opening from one end to allow the loader to climb up on the pile, if necessary, for inspection, or to work the ore down. A 3/4-in. pipe is laid in the concrete lining, to serve as a speaking tube between the loader and the trammer above.

The ore from this stope had formerly been dropped from the 200 to the 400 through a standard, timbered chute. No attempt was made

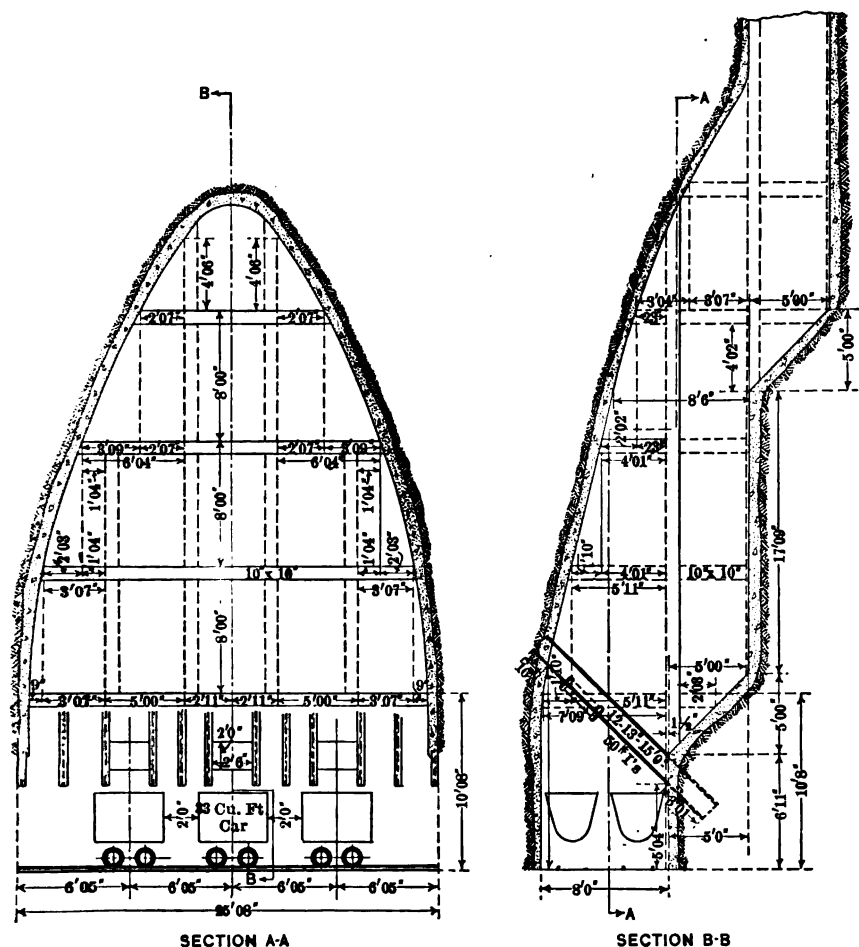


FIG. 6.—CZAR ORE POCKET.

to use the chute for storage, and the door at the 400 was kept open by the loader, but other men were required above to keep the chute clear and the ore moving.

It was found that the whole capacity of the storage bin could not be used, the weight of the ore packing it too hard; but if the quantity

does not exceed 50 or 60 tons, the loading is rapid, and there has been no accumulation of ore in the concrete chimney at any time. In rebuilding it, the storage chamber would not be made so high.

The timbered chute shown in Fig. 7 is sometimes used successfully for stopes. The half bulkheads partly support the ore, and it does not pack so badly in the bottom. It was used by the Cananea Consolidated Copper Co.

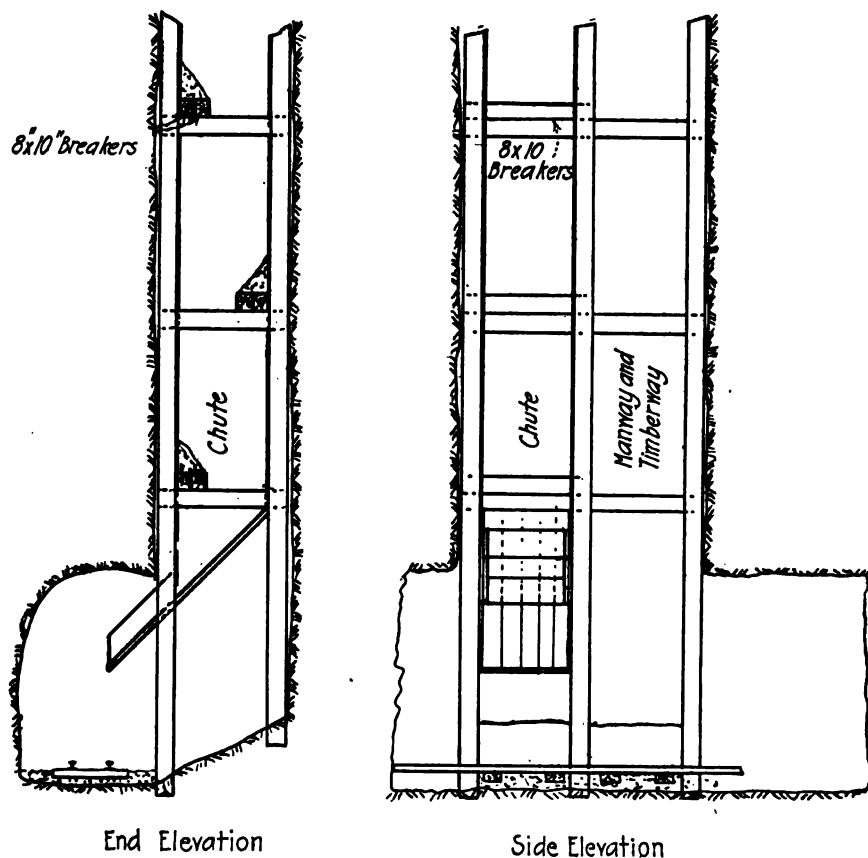


FIG. 7.—TIMBERED CHUTE.

Sacramento Selected for Hoisting Shaft

The greatest month's production prior to July, 1907, had been, 55,347 dry tons, in January, 1906, or 1,910 tons per working day. It was not exceeded until March, 1912. The desired capacity of the plant was assumed to be 60,000 tons per month, or about 2,150 tons per working day, with a reasonable margin for emergencies. At that time, the mine

worked alternate Sundays, and three shifts were run in tramming and hoisting.

The Sacramento shaft had been sunk for general development purposes, and had been located within the porphyry intrusion, to avoid the ore zone and its possible movement. It was selected for the hoisting shaft since it was in a central position and convenient to the main line of the railroad.

It was 928 ft. deep to the 1,000 level, and had three compartments 4 ft. 6 in. by 5 ft., and a half compartment for air pipes, etc. Since then, it has been sunk to the 1,700 level and widened to five compartments, making two for skips, two for cages and a large one 5 ft. by 5 ft. for pipes and cables.

It was at once decided to use skips.

The size of the compartments restricted the cross-section of the skip to 4 ft. 2 in. by 3 ft. 3 in. inside. The effective capacity fixed upon was 60 cu. ft. It was thought that one of large capacity would be too long, and that the bottom layer of ore might be packed so hard by the fall into the skip and the weight of a high column of ore upon it, that it might stick. This has sometimes been the case with the size adopted. The average skip load in April, 1915, was 8,117 lb. of wet, or 7,459 lb. of dry ore. The moisture, 8.11 per cent., is the lowest in years, the average would be nearer 12 per cent.

There would be no difficulty in raising much more than the quantity of ore assumed, when it runs freely, even with small skips. When it does not, large skips are of no particular advantage.

The uncertain factor was the rate at which ore could be drawn from the storage bins, which are necessary when skips are used. The experience with bins had not been particularly successful. Attention was therefore concentrated upon providing facilities for rapid loading, and to give loaders the maximum time for work at the loading door.

The size of the skips, power and efficiency of the hoist and economy of rope wear are interdependent. Conditions favorable to one are usually opposed to efficiency in the other.

A larger skip requires a more powerful and expensive engine, the efficiency of which is less, due to less continuous hoisting for the assumed tonnage, and greater condensation. There might be a small saving in rope expense, as the skips would make fewer trips.

Maximum efficiency in ropes would be obtained by larger drums, or drums long enough to avoid two layers. Larger drums require larger cylinders and again an engine more costly and less efficient. There is no great advantage in the use of larger drums unless they are large enough to hold the rope in one layer.

The capacity of the skips, the proportions of the engine, and diameter of the ropes were arrived at by one of those compromises, which must be

so frequently made in balancing an advantage against its complementary disadvantage.

Reports of the steam economy of compound engines operating in South Africa had been received, and after thorough investigation by John Langton, Consulting Mechanical Engineer for the company, a duplex, tandem-compound, condensing, first-motion engine was purchased from the Nordberg Manufacturing Co. The conditions were not the most favorable for a compound condensing engine, as the depth of the center of gravity of production in 1906 was only 570 ft., but it was expected that the depth of hoisting would increase. The center of gravity of production is now 970 ft. below the collar.

The plant is driven by steam at 150 lb. power-house pressure, transmitted 619 ft. through a 5-in. main. The steam leaves the power house with 75° F. of superheat. The engine can lift an unbalanced load of 17,000 lb. Its speed is rated at 2,000 ft. per minute. The reels hold 2,100 ft. of 1½-in. rope in two layers. The cylinders are 18 in. and 28 in. in diameter by 48-in. stroke.

Most of the details are of standard Nordberg design, but there are three unusual features:

The operation of the auxiliaries.

The regulation of pressure on the low-pressure cylinders.

The condensing system.

Auxiliaries Actuated by Oil under 135 lb. Pressure.

When steam is used by the auxiliaries, which operate brakes, clutches, etc., it constitutes an important percentage of the total power used by the engine. The condensation is great and continuous, and since the controlling valves can have no lap, but must be set "line and line," it is impossible to avoid leakage as well. The steam consumption by auxiliaries in a simple engine running at capacity is rarely less than 14 per cent. of the total. If the engine runs intermittently, the proportion used by the auxiliaries is correspondingly greater. In a more efficient engine, the steam for auxiliaries remains the same and is also a greater proportion of the whole.

In this engine, throttle, clutches, brakes and reverse are actuated by oil under a pressure of 135 lb. and controlled by poppet valves. The oil discharged from the auxiliaries is pumped back into an accumulator by a triplex pump, with plungers $2\frac{7}{16}$ in. in diameter by 4-in. stroke, which are driven by eccentrics on the valve-gear shaft. When the accumulator plunger is raised to its highest position, the oil is bypassed to a reservoir. When hoisting from the upper levels, the pump is too small and a relay steam pump which cuts in when the accumulator plunger is below a

certain point runs most of the time. This method is entirely satisfactory, and the engine control is sensitive, positive, and efficient.

The operating levers are arranged in a standard manner (Fig. 8) first used on the Frazer and Chalmers 20 by 48 in. engine at the Spray shaft. All the large hoists are controlled by levers on this plan. There is a great advantage in having all engine levers in relatively the same posi-

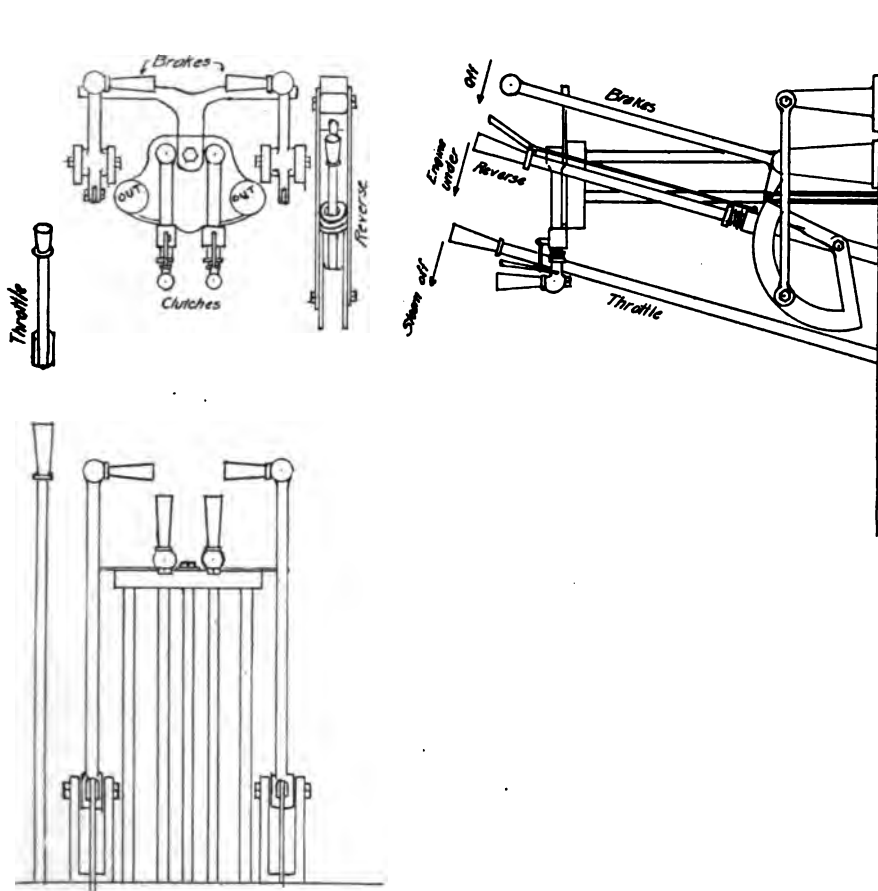


FIG. 8.—TYPICAL HOIST-LEVER OPERATING SYSTEM.

tion, acting by the same motion. Engineers can be transferred from one hoist to another without danger of confusion.

Pressure on Low-pressure Cylinder Automatically Regulated

A large steam-jacketed receiver is placed between the high- and low-pressure cylinders, in which pressure is maintained at 20 lb. by an automatic valve, through which steam is bypassed from the high-pressure

main to the receiver, when the throttle is closed. Thus, the engine has pressure on all cylinders when starting, and picks up smoothly and quickly instead of needing a stroke or two to reach a working pressure in the low-pressure cylinders. When the throttle is opened, the bypass is shut off and the pressure in the receiver drops to that dependent on the load, speed, etc. When it is desired to hoist with great speed, quicker acceleration may be obtained by adjusting the reducing valve to raise the receiver pressure.

Condenser Operates under Severe Conditions

The engine has Corliss valves, with automatic cutoff regulated by a governor. Severe conditions are imposed on the condenser by the irregular use of steam. During the first two or three strokes, while the engine is starting the load, steam is taken nearly full stroke, while at the end of the trip the cutoff may be at $\frac{1}{10}$ stroke. In order to condense this sudden rush of steam, a large reservoir of cold water is provided in the "Rain Type" Nordberg condenser cylinder, into which the steam is exhausted, by a series of shallow flat trays filled with water. Constant streams of water are poured into them which over-flow, rain down to the bottom of the cylinder and finally run off by gravity against the vacuum, to the cooling tower, from which it is re-pumped. A 900-gal. rotary pump is used for circulating the water and a 19 by 18 in. vacuum pump, both belted to 20-h.p. 220-volt alternating-current motors which run at constant speed. The temperature of the water at present (June, 1915), is 84° F. when entering and 96° F. when leaving the condenser. The barometer stands at approximately 25 in. The vacuum drops to 16½ or 17 in. at the beginning of a trip, but rises to 20 or 20½ in. The man engine also runs condensing and exhausts into the same system, which was increased in capacity by large pumps and a second condenser when the man engine was added.

Hoisting Speed 1,300 to 1,600 ft. per Minute

In order to insure continuity of operation under all conditions, the low-pressure cylinders are built to stand high-pressure steam. The receiver pressure can be built up to 150 lb. and the engine operated as a simple engine with cylinders 28 by 48 in. stroke. This makes it an engine of great power for emergency use, as for instance, hoisting unbalanced, but at a sacrifice of efficiency.

The engine governor is set for a rope speed of 1,300 ft. per minute when hoisting from the 1000 level or a level above, but by shifting the governor belt to another pulley, a speed of 1,600 ft. is used for greater depths. When production is increased, both speeds are raised. It is

preferred to run the hoist at as regular and slow a rate as will conveniently hoist the ore.

Round Rope on Grooved Drums Now Used

Round ropes, $1\frac{1}{4}$ in. in diameter, in two laps are used on reels, which are 7 ft. in diameter by 5 ft. face. The drums were made without grooving at first, as it was uncertain what ropes were best, a high-grade rope $1\frac{1}{8}$ in. in diameter or $1\frac{1}{4}$ in. rope of less tensile strength. The ropes began to cut into the drum in an irregular curve. They were grooved in April, 1914, to a depth of $\frac{5}{16}$ in., leaving the diameter of the drums at the bottom of the grooves $83\frac{3}{8}$ in.

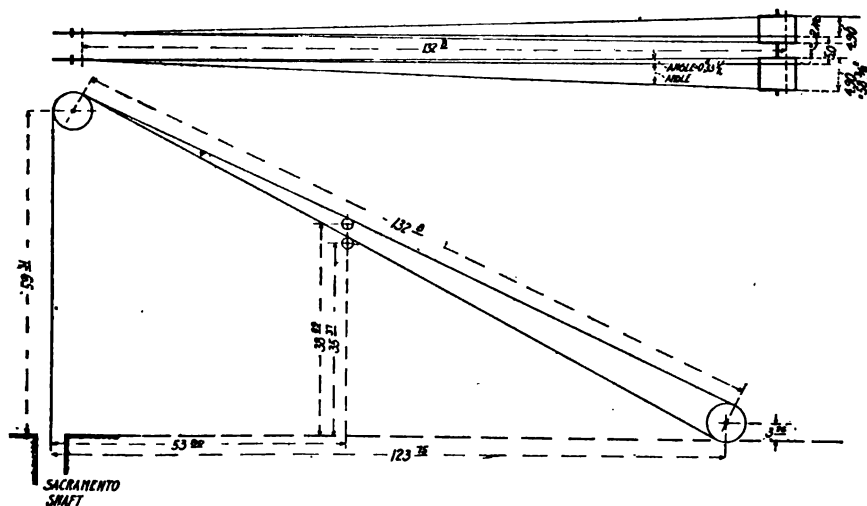


FIG. 9.—LEAD AND FLEET ANGLES, SACRAMENTO ROPES.

Fig. 9 shows the relative positions of drums and sheaves to shaft. The ground back of the shaft was cut out as far as practicable to get a reasonably long lead from the sheaves.

The fleet angles are unequal, and from the sheave line to the outside of the drum is $1^{\circ} 33'$. This would be considered excessive in certain districts, the copper country of Michigan for instance, but accepted practice in other places permits an angle of $1^{\circ} 55'$. The original attachments of the ropes were at the outside, or clutch side, of the drums. There was sufficient angularity in the pull from the sheave to draw the laps apart and leave openings into which the second layer of rope dropped. The winding was irregular and rough, and the wear excessive. Other sockets for rope attachment were cut on the inside of the drums, and the trouble ceased. The angular pull tends to hold the laps together in-

stead of separating them. When the drums were grooved, the ropes were again attached at the outside, since the active part of the rope would be nearest the sheave line, and the fleet angle smaller. There is heavy side friction on the rope when the last laps of the second layer are wound on. If attached at the inside of the drum, the grooving would carry it to the outside without so much friction as if winding against a rope. On the second layer, the pull of the rope would again tend to separate the laps and reduce the side friction of rope against rope. Soft-steel wedge-shaped fillers lift the rope out of the inside groove, high enough to wind back on the first layer.

No difference has been noticed in the wear of the under or over wind. If there is a reasonable distance between sheave and drum, there should be none.

The sheaves are 7 ft. in diameter, and are lined with leather. Wood lining broke down rapidly. Cast-steel idlers support the ropes between sheaves and drums to reduce slapping. They are cast thin and the flanges machined to make them still thinner and lighter.

The ropes are attached to the skips by thimbles and six Crosby rope clamps. Once every four weeks, 6 or 8 ft. is cut off the end of each rope to change the points of maximum wear.

The shaft is usually dry. In the wet season, a little acid water seeps in, but no corrosion has been discovered.

A few years ago, Mr. Duncan, Agent of the Cleveland-Cliffs Iron Co., kindly furnished rope records from the Lake shaft hoist, which has 7-ft. drums, 9-ft. sheaves and uses 3½-ton skips, but with only one layer of rope on the drum.

TABLE III.—*Data on Hoisting Ropes*

	Cleveland-Cliffs	Copper Queen
Diameter of drum, feet.....	7	7
Diameter of sheave, feet.....	9	7
Length of rope, feet.....	1,800	4,000 (2 ropes)
Diameter of rope, inches.....	1¼	1¼
Weight of skip, pounds.....	4,000	6,350
Weight of ore, pounds.....	7,000	7,400 (dry)
Weight of rope, maximum, pounds.....	1,347	3,880
Ore and rock hoisted, tons.....	700,000	395,569 (dry)
Depth of average hoist, feet.....	550	970
Depth of maximum hoist, feet.....	550	1,550

The Copper Queen rope averages do not include those ropes which were obviously unsuited to the conditions.

The ratio of rope to drum diameter is 1:67.2. The experience of the Cleveland-Cliffs Iron Co. indicates that good service may be obtained with that ratio.

The rope cost on the Copper Queen ropes in the above comparison was 0.3 c. per dry ton. The ore contained about 11 per cent. moisture during the time covered by the records.

From the above comparison, the life of a rope is increased 75 per cent. by the use of one lap instead of two, and the cost for ropes at the Sacramento shaft would have been in consequence, 0.17 c. instead of 0.3 c. This would represent a saving in rope in an average year of about \$1,000. In order to obtain this saving, the cost of the engine would have been approximately \$15,000 greater, and it would be less efficient. A geared hoist would avoid some of these objections, but would introduce others.

The greatest tonnage, 549,324 dry tons, was hoisted by a pair of flattened strand ropes with Lang lay. The cost was 0.277 c. per dry ton. The increase in wearing surface provided by flattened strands is very noticeable. In winding on a smooth drum, the contact between two consecutive laps is made by a rubbing motion. In the ordinary lay, the outside wires which touch are parallel with the length of the rope at the point of contact, and the rough surfaces of the projecting strands bear against each other when the rope comes down against the last wrap. With the Lang lay, they are inclined to the line of the rope and each other, and the friction is reduced.

These conditions are of paramount importance when two layers are wound on a drum, since the bearing of the second lap is entirely rope against rope, and there is at the best a good deal of sliding of one rope on another. The external wear only need be considered, internal wear is of no consequence in comparison. Efficient lubrication on the outside of the rope is of great benefit.

This question is treated in some detail, because it was found very difficult to obtain information on the effect of small drums and two layers of rope on the cost of hoisting. It was of course recognized that the wear would be increased, but to what degree it would increase the cost could not be ascertained.

Underground Storage Bins Have Loading Hoppers

The storage bins at the haulage levels were at first made of standard design. The only attempt to avoid the difficulties formerly experienced with bins was to make them comparatively small to avoid pressure.

Attention was particularly directed toward giving the loader as much time as possible to poke the chute. This was done by putting measuring hoppers holding one skip load each below the bins, which could be loaded at any time while the skip was in motion and tripped into the skip when it reached the loading position. When hoisting in balance at the maximum rate expected, a skip per minute, and allowing 15 sec. for

tripping the hopper and ringing off the skip, each loader has $1\frac{3}{4}$ min. to load the hopper.

When starting the plant, it was found at once that the customary bin was not satisfactory. Not more than half the requisite speed could be attained. The 800-level bin was altered to one of the present type, but of small size, 40 tons capacity. As soon as it was found successful, the 400 and 600 bins were rebuilt but on a larger scale. Their capacity is 90 and 100 tons. The three lower bins are about the same size.

The bin as reconstructed consists of two parts, an upper chamber for storage and a lower part open in front which acts as a slide to the gates. The horizontal opening at the bottom of the storage chamber is 4 by 8 ft., large enough to prevent arching, but small enough to carry the weight of the ore in storage without permitting it to exert much pressure on the ore resting against the gate. The front, immediately below the contraction, is open for an added relief against pressure as the ore is free to take its natural slope.

This principle has since been used in a 260-ton bin for the 400 level, which is shown in Figs. 10 and 11, together with a standard wooden bin. In this case, the excavation was made circular and lined with concrete lightly reinforced. The conical bottom is armored with rails set in the concrete. It has been found necessary to protect the concrete against erosion when hard ore is stored, but only where the ore strikes on rebounding from the inclined plane at the bottom of the chimney.

It was not known at first how large the hole in the bottom of the cone should be. It was accordingly made 6 ft. in diameter, with the expectation of enlarging it if necessary. It has since been cut out to 7 ft. 4 in., which appears to be the correct proportion to protect the ore below from excessive weight and still prevent arching. This storage bin, although much larger than the others, handles muddy ore equally well.

As finally constructed, the large bin was built on the opposite side of the shaft from that shown in the drawing.

Arc doors are generally used, 30 by 16 in. in size, with good results. The slight extra labor of operation is balanced against the higher cost for installation and maintenance of power doors, which are only used at the large concrete bin at the 400 level.

Below the doors are the loading hoppers (Fig. 12) which hold 58 cu. ft. each, or one skip load. The flat counter-weighted bottom is hinged at the rear and held shut by tension bars on each side extending up to a shaft which is turned by hand levers. It dumps quickly and locks positively, as the tension bars are turned past the center of the shaft. The chute beneath, which carries the ore to the skip, is very steep, to prevent sticking.

The operation of loading is very rapid under ordinary conditions.

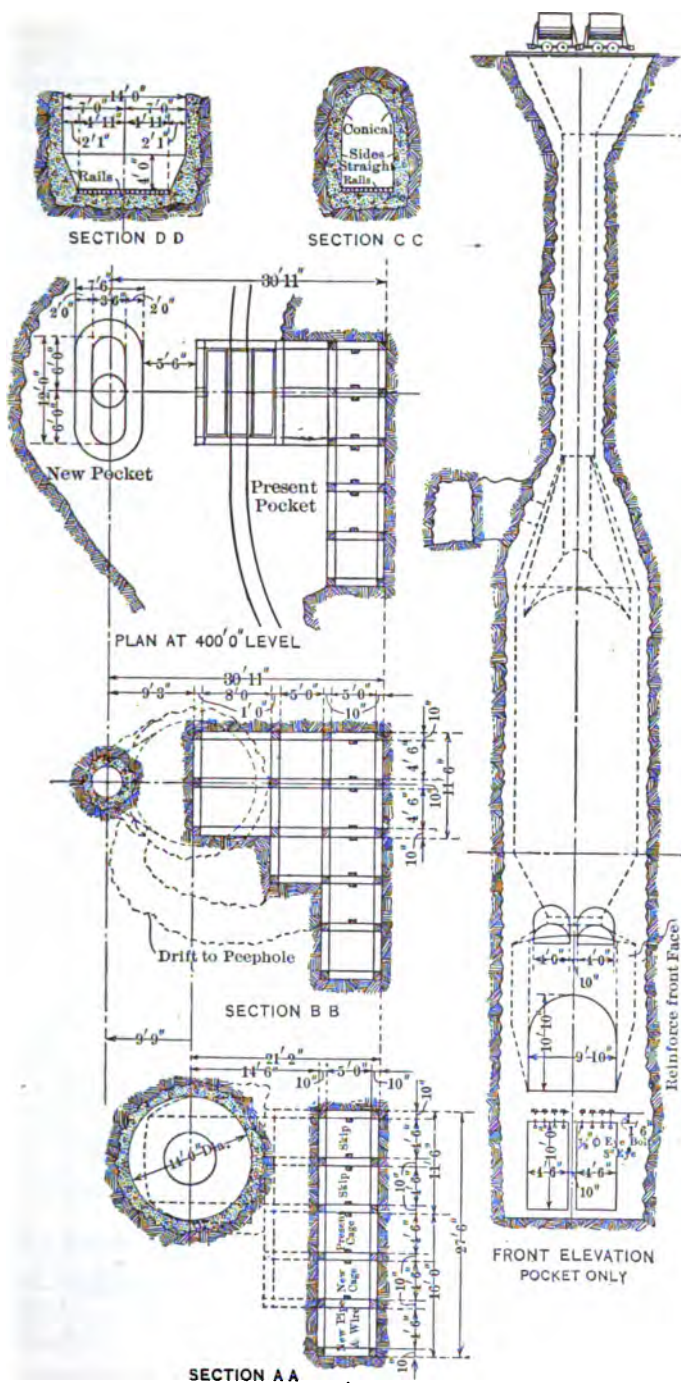


FIG. 10.—260-TON ORE POCKET, 400 LEVEL.

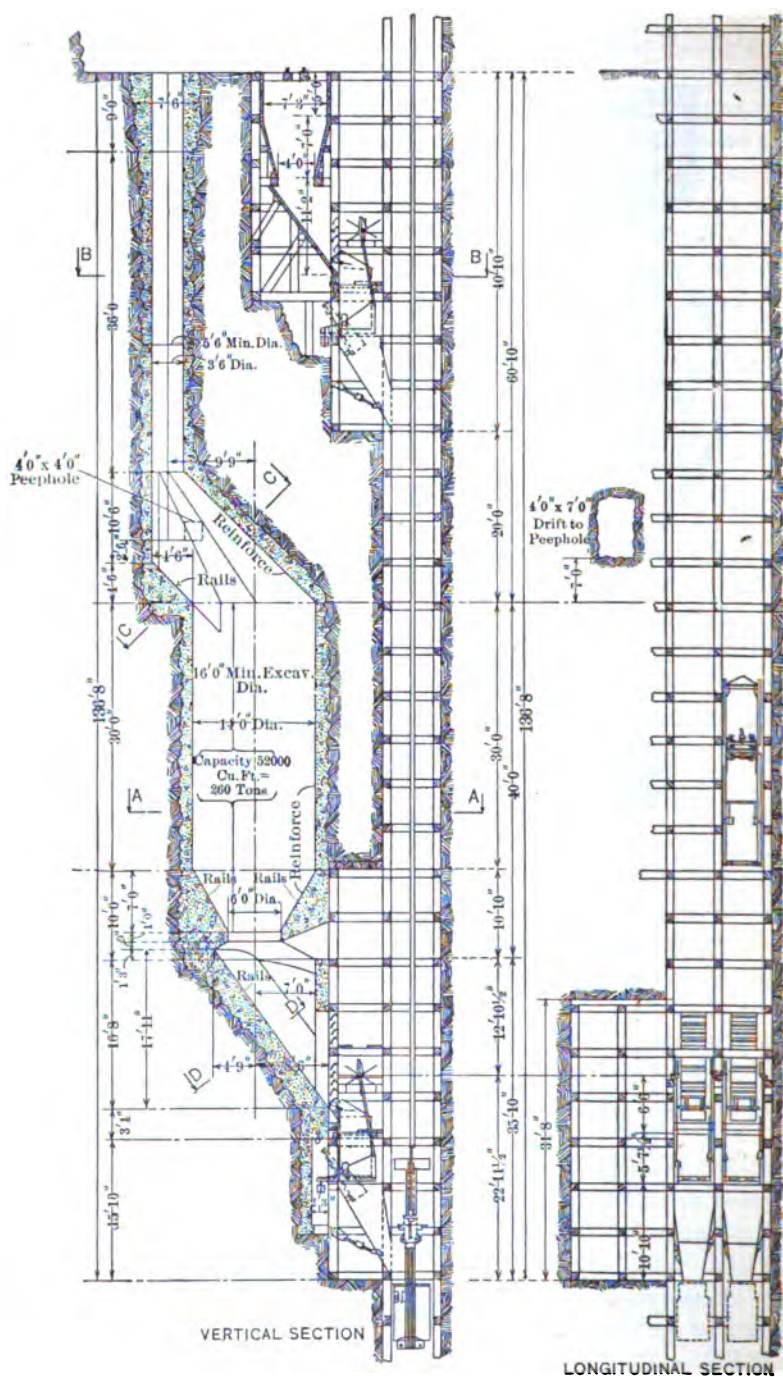


FIG. 11.—280-TON ORE POCKET, 400 LEVEL.

The time required to dump the ore into the skip and ring it off rarely exceeds 12 sec., and frequently not more than 10 sec.

The best day's record is 3,899 wet tons in three shifts, or 3,467 dry tons. An average has been maintained for several shifts at the rate of

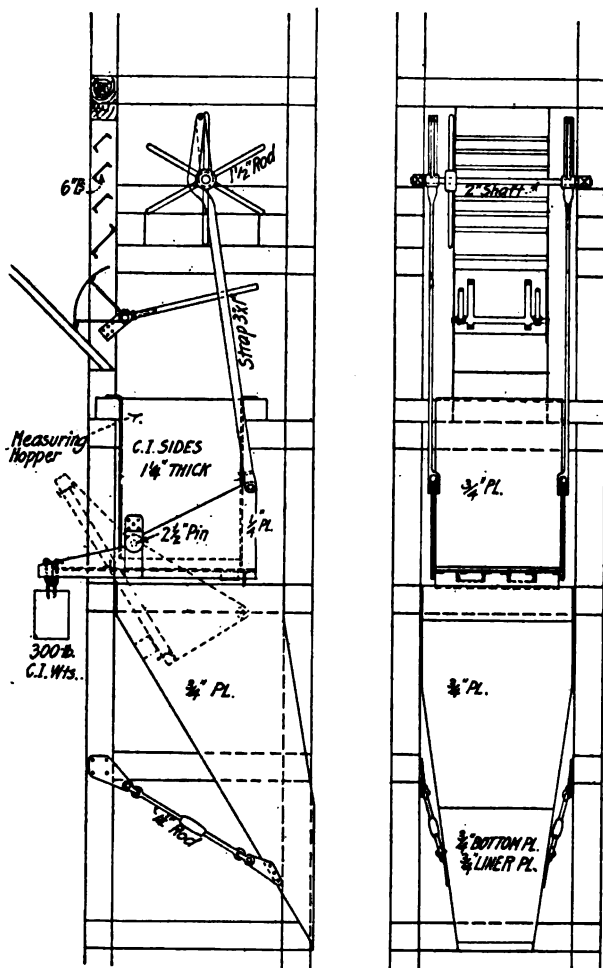


FIG. 12.—SACRAMENTO MEASURING HOPPER.

433 skips per shift. When the 30 min. for lunch and the time estimated for changing levels are deducted, it makes a hoisting rate of 1.1 skip per minute. Two loaders are required. When there is much muddy ore, or when hoisting very rapidly, a third man is sometimes used. There are still occasions when, for short periods, not more than 20 or 30 skips per hour can be hoisted.

The construction of skips and loading mechanism is heavy and strong. There have been few delays and light repairs.

There has been trouble with the spillage of ore while loading. The chute through which the ore reaches the skip is so steep that the opening is 40 in. high. Even if the skip is held only 3 in. below the bottom of the chute when spotted for loading, there is an opportunity for the ore to spread in falling, and for some of it to fall on each side of the skip rather than in it.

A screen was attached to the side bars above the skip, composed of light plates, inclosing three sides but open on the loading and dumping side. It reached down to the top of the skip, but was spaced far enough outside, so that the skip could turn through it to dump. Since there was a certain amount of danger in adding this complication to the skip, and the spillage was not greatly reduced, it was removed.

The spilled ore falls into a pocket at the bottom of the shaft, from which it is loaded into cars and hoisted on the cage to the 16th level bin. It amounts to about 17 lb. per skip, or 0.21 per cent. of the ore hoisted. It results partly from ore falling outside the skip when loading, partly from the ore which hangs up in the slide below the loading hopper and which falls after the skip is hoisted and partly from overloading. If the skip does not dump clean, and it is not noticed, there may be enough remaining to make an overload when the next charge is tripped into it. As a result of the spill, there has been heavy wear on the timber in the hoisting compartments.

The skips had originally only 70 cu. ft. capacity, and not sufficient margin to prevent an occasional overload. They were rebuilt and made to hold 90 cu. ft.

The skip is of standard Kimberly type, but built heavily. The dumping guides on the surface are formed of angles and heavy steel castings. As the castings wear, thin lining plates are riveted on.

Ore Loaded into Railroad Cars by Belt Conveyors and Trippers

The ore is dumped into a small bin holding about 12 tons, from which it is carried to the railroad track below, and loaded into cars for shipment to the reduction works. The arrangement of the conveyor plant is shown in Figs. 13 and 14.

A 36-in. steel pan conveyor forms the bottom of the bin, and in transporting the ore to an inclined belt equalizes the load. The receiving bin was made small to avoid a great accumulation of ore and consequent packing. It is now apparent that a larger bin would have been permissible and desirable.

The ore falls from the pan conveyor into a belt loader which breaks its fall and feeds it to the belt in the direction of belt travel (Figs. 15 and 16).

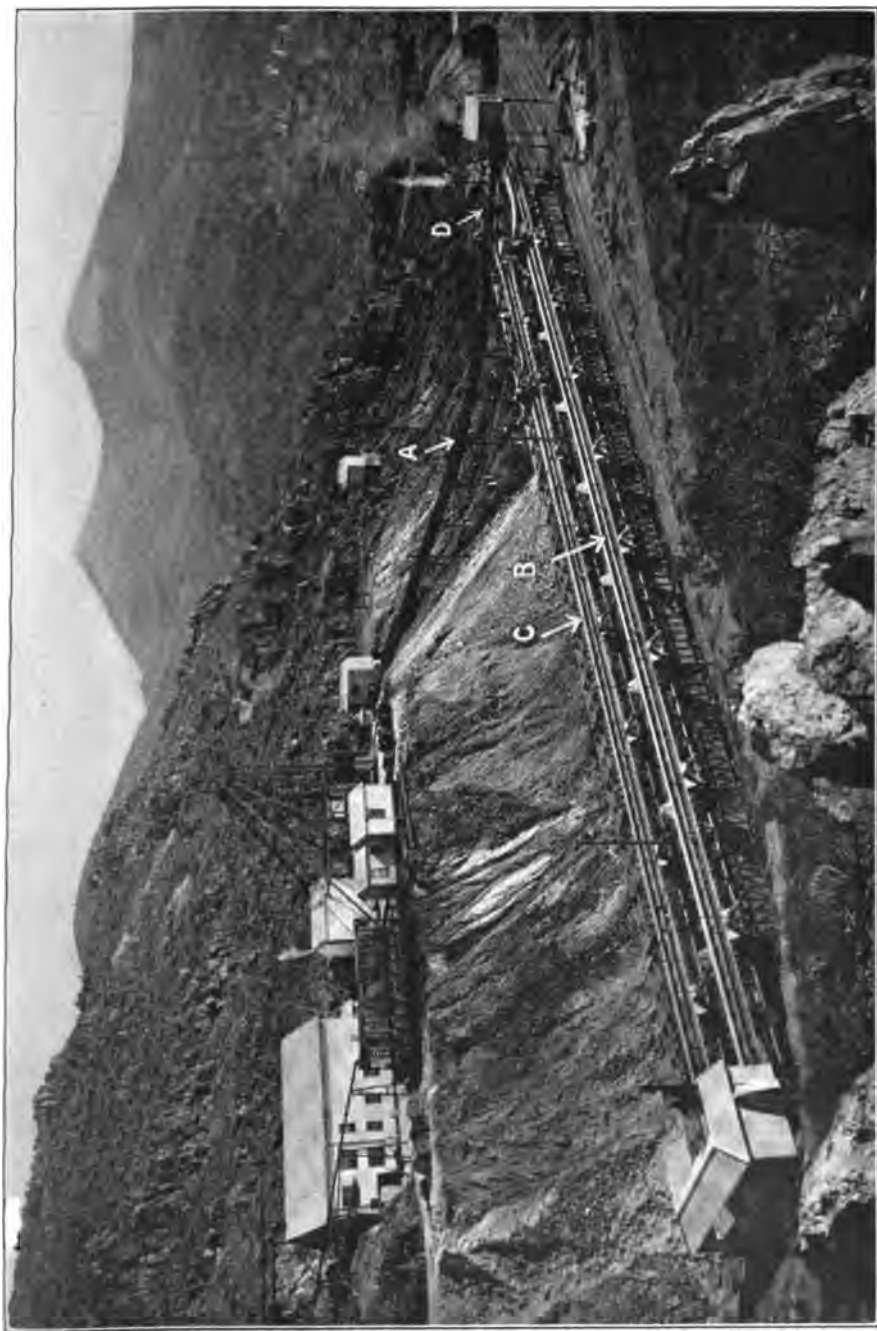


FIG. 13.—SACRAMENTO SHAFT LOADING PLANT IN 1909.

The V-shaped opening at the bottom of the pan allows the fine material to reach the belt first and form a bed for the coarser. The sides of the "V" are movable, and may be spread apart to widen the opening when sticky ore is hoisted.

Great difficulty was encountered in obtaining a feeder which would not clog with clayey ores. Even as it is, a solid mass of clay has accumulated above the plunger, the full width of the feeder, and several feet high, with the plunger still running ineffectively behind.

When the conveyor plant was first built, the ore fell from the pan conveyor upon the unprotected belt, which travelled on a slope of 15°. Boulders of ore sometimes bounded down the incline before they came to rest on a bed of finer material. When the feeder was put in, the upper 18 ft. of the belt was depressed to a horizontal position, and the bed of ore is well settled before it reaches the incline.

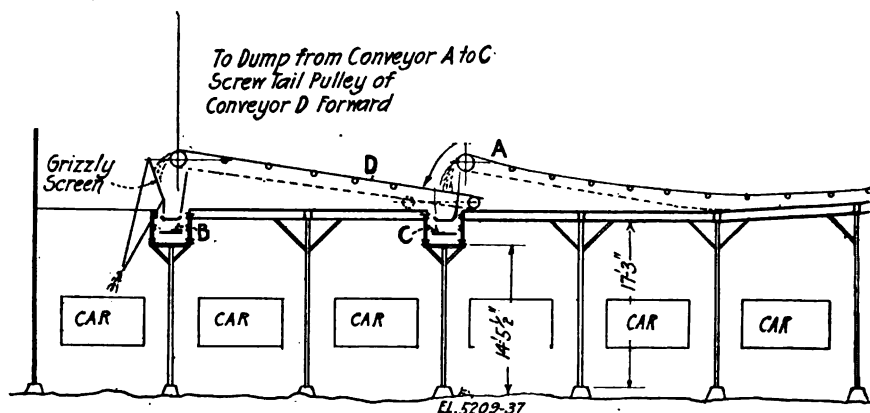


FIG. 14.—PORTION OF SACRAMENTO CONVEYOR SYSTEM.

The inclined belt A (Figs. 13 and 14) carries the ore down to the loading tracks, passing over a Blake-Dennison weighing machine on its way.

It is supported in a catenary curve to reduce the height of the trestle. This is now recognized as a mistake, as the maximum inclination is increased and the belt cannot be run taut without lifting it off the rollers when it is run empty, or with a light load.

The line of the incline belt crosses four railway tracks and delivers ore to conveyor C, the first of two conveyors parallel with them. The ore is discharged off the end of the belt and falls into a belt feeder similar to that at the shaft. When the ore is to be carried to belt B, the end pulley of the short belt D, the bearings of which are supported on slides, is screwed back by hand under the stream of ore discharged from A. It is then carried by the short conveyor to B and dumped into the third



FIG. 15.—BELT FEEDER.

belt feeder. The short belt thus takes the place of a tripper on a continuous belt and reduces the belt cost.

The loading belts *B* and *C* are each located between two tracks, and are long enough to load seven 50-ton cars and a sample car.

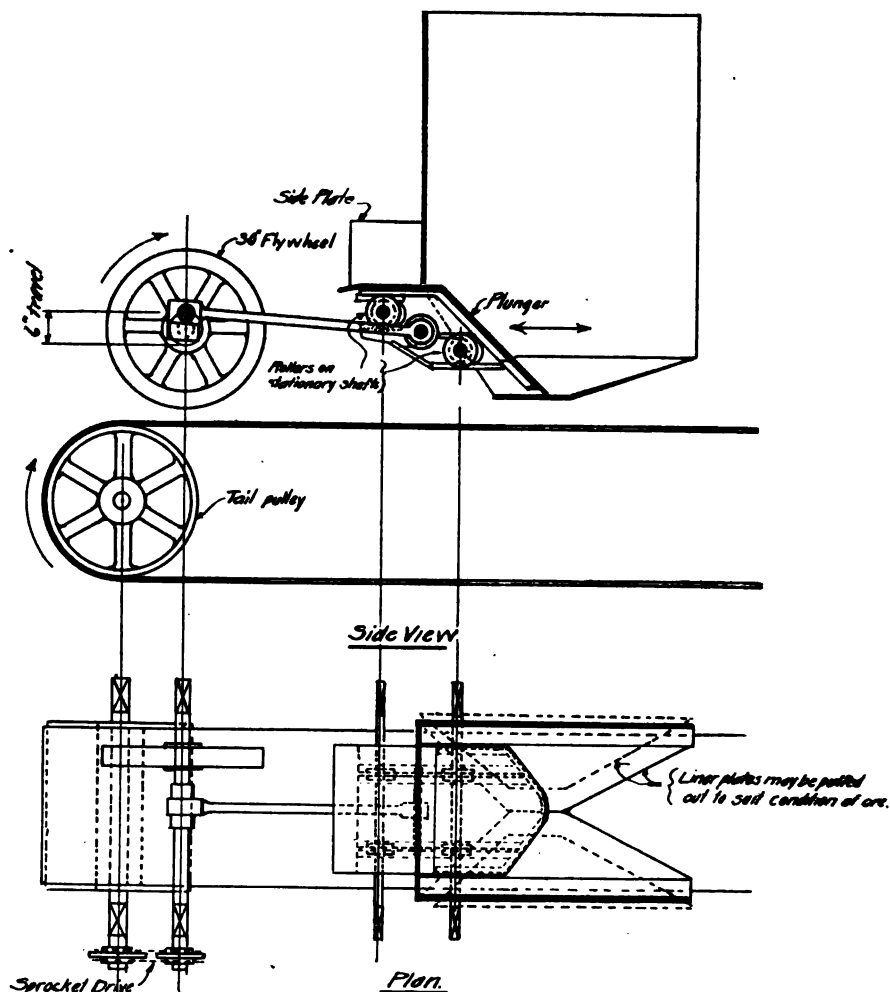


FIG. 16.—CONVEYOR PLUNGER FEED.

A tripper (Fig. 17) moves back and forth constantly, depositing layer after layer of ore in the cars below, until they are loaded to capacity as shown by the belt scales. At one end, it travels a few feet over the eighth car and this is taken as a sample of the lot. When the first lot is full, the sample car is switched to the other end of the loading shed and attached to a string of seven cars loading from the other conveyor,



FIG. 17.—BELT TRIPPER.

and the sample of another 350 tons is dropped into its opposite end. Its sample therefore represents two lots of ore of 350 tons each.

The belt speeds may be varied from 240 to 360 ft. per minute, but rarely exceeds 300. The tripper speed is from 150 to 190 ft. per minute, but at the maximum speed of the belt would be 225. It travels back and forth from 36 to 40 times in filling a lot, depending on the weight of the ore; thus depositing from 72 to 80 layers of ore in the cars, and taking half as many samples while doing so. Although each lot may be of widely different composition, the ore in each car or in each linear foot of the car is the same as any other in the same lot. At the smelting plant, the ore is dumped into pits, one lot directly on the other, until a bed of 10,000 tons is made up. It has been found that when loading by steam shovel from one end of this bed, the ore encountered in working through to the other end is sufficiently uniform in composition to secure good metallurgical results. The sampling is also sufficiently accurate.

In its travel, the stream of ore from the tripper crosses a spout 12 in. wide, which catches a sample, splits it, returning one-half to the car, and drops the remainder into a heap on the ground, which is quartered down and assayed for quick returns. It is surprising how closely this extremely rough sample checks with the smelter returns. Although there is considerable variation in the sample of individual lots, the average for a month's run is rarely more than 0.1 per cent.

The first spout tripper was not positive in operation, and a man was needed to ride on it to keep it clear when loading mud. A special tripper was built, in which the ore is discharged upon a short 36-in. belt running at right angles to the main belt. It is driven by a small reversible motor carried on the tripper, which collects its power from a trolley wire, and may load into either of the string of cars spotted below. The spout tripper was built so high, to secure a steep incline for the spout, that it was unsteady and it was impossible to prevent the main belt from running out of line and cutting the edges. The belt tripper is low and steady and the main belt runs more truly and with less wear.

The tripper belts cost 0.266 c. per ton of ore handled. Now that there is a smaller proportion of muddy ore, a low heavy spout tripper might be better. Less attention than before would be needed to keep the spouts clear.

The inclined belt *A* is driven by a 30-h.p. direct-current variable-speed motor at the lower end. The belt travel can be varied between 240 and 360 ft. per minute. The pan conveyor at the shaft is driven from the upper pulley by a sprocket chain, as is the belt feeder also. Belt *D* is also driven from this motor, through a friction clutch. Very little power is required, except to start, about $10\frac{1}{2}$ h.p. with, and 8 h.p. without belt *D*. When loaded and in motion, it will drift so far that it is

equipped with a brake in order to stop it quickly when necessary. The two loading belts *B* and *C* are driven by a 30-h.p. variable-speed direct-current motor at the far end, through friction clutches. The tripper belts are driven by 5-h.p. motors.

The trippers are pulled back and forth by $\frac{1}{2}$ -in. steel cables driven through reversing clutches, geared to the drive shafts of the belt. Buttons on the rope throw the clutches over at points of reversal, by turning the three-way valve of a compressed-air cylinder, the piston of which acts on the clutch levers. The buttons and points of reversal may be changed quickly, but there is little alteration in the routine and it is rarely necessary. The operation is positive and very satisfactory since powerful clutches have replaced the lighter ones formerly used. The conveyor belt passes idly through the tripper, and there should be less wear than if it supplied the power to propel it.

The control of the whole plant is located on a platform between the loading belts and near the incline. Here are the starting switches for all motors, brake for the incline belt, and levers connected by cables with the clutches at the driving end of the belt conveyors. The belt speed is governed by the speed of hoisting. They are run faster when hoisting from the upper levels. They are stopped when the skips are delayed or stopped for any purpose. The conveyor man is stationed at this point, where he can see all the belts and everything that goes on. One Mexican helper oils up rollers, chains, pulleys, etc. Another is stationed at the shaft to look after the loading end. A mechanic on day shift has been found desirable to attend to current repairs, to re-lace belts, etc. Other men are required from time to time to make more extensive repairs, clean tracks, attend to sampling, etc.

Many changes have been made in the original installation. Belt feeders have been added. The incline belt has been cut in two to eliminate a tripper. The trippers have been rebuilt. The control of the driving mechanism has been centralized. A roof has also been added to protect it from the weather.

The ore is rough, uncrushed, and very hard on belts. The life of belts is gradually increasing, due to improvements in the plant and a better understanding of the conditions.

Recently, it has been desired to screen a certain class of ore for fettling in the reverberatory furnaces, and to supply a coarse sulphide for pyritic smelting. This has been done by raising the discharge end of belt *D* so that the ore falls against a screen above the belt feeder. The fine ore falls into cars directly below, the reject into the belt feeder and makes up a regular lot and sample of coarse screened ore. Sufficient for present needs can be screened out in this manner, but if more were required, more costly changes would be needed.

Increase in Underground Storage Capacity Contemplated

After the changes in the shaft storage bins, no others were necessary to reach the capacity desired or increase the certainty of operation. Alterations have been for purposes of economy or convenience. There have been few delays caused by accident, and none which stopped operations for more than a few hours.

As stated above, the most rapid hoisting in one day was 998 skips, containing 3,467 dry tons of ore, in three shifts of $7\frac{1}{2}$ hours net working time each. There was not enough ore to fill out the third shift at the same rate as the other two. The rate for an hour or two has exceeded this considerably. The maximum rate for any month was in February, 1914, when 65,901 dry tons were hoisted in 58 shifts, or 1,136 per shift. When hoisting at this rate, the shaft bins have not sufficient storage capacity. Time is wasted in changing frequently from level to level to avoid blocking the haulage motors. It is to prevent this that the 260-ton bin on the 400 was put in. Two others are in contemplation, which would increase the storage capacity to 1,350 tons. The hoisting capacity of the shaft would be increased, and there would be fewer delays to electric haulage.

Compressed-Air Hoists for Men and Timber at Most Shafts

The engine for men and timbers at the Sacramento shaft is a compound, two-cylinder engine, made by the Nordberg Manufacturing Co., but drives through Wuest herringbone gears. The cylinders are 18 in. and 30 in. by 42 in. stroke, the gears are 32 in. and 73 in. in diameter, with drums 9 ft. in diameter by 48-in. face, grooved for $1\frac{1}{4}$ -in. rope. The brakes, clutches, etc. are actuated by oil under pressure from the same accumulator which the ore hoist uses, and in a similar manner. The two triplex pumps to supply pressure for the auxiliaries have plungers $2\frac{1}{2}$ in. in diameter by 6 in. stroke, but the oil used when landing several decks is more than it can supply. Three-deck cages in balance are used.

When ore hoisting was transferred to the Sacramento shaft, the Holbrook, Spray and Lowell steam hoists were connected up to drive by compressed air from the central power plant.

The Gardner hoist was at first driven by steam from the power house, delivered through a 5-in. main, 920 ft. long. Upon making tests on the efficiency of hoisting by compressed air, the latter was found to be more economical, and it is now driven by air also.

No changes were made in the engines when using compressed air. The cylinders for brakes, clutches, etc. use cold air. The leakage has been less than when hot air was used and there has been less trouble with packing.

The air for the cylinders is heated to a temperature of 225° to 275° F. The cylinder of the heater is filled with heavy high-test oil, which flashes at about 700° F., and is inclosed in a brick setting like a small boiler. It is heated by a small coke fire under the front part, which is covered by a brick arch to protect the oil from overheating. The gases return to the stack in front through fire tubes in the lower third of the cylinder.

The air makes three passes through the cylinder in small tubes. An open pipe from the dome carries off any products of distillation. The temperature of the oil ranges from 250° F. to 275° F., although it may be carried higher without undue loss of its lighter elements.

The reheater supplies a reservoir of heat for the heavy drain on the system when the engine is starting, and does not over-heat if the engine is idle for a short time.

At the Lowell shaft, the air is heated to an average temperature of 250°. The air consumption there is estimated to be 30,000,000 cu. ft. of free air per month, at a gauge pressure of 92 lb. In the past five months, the consumption of coke has been 3.8 tons per month.

The oil grows slowly thicker by the gradual loss of its volatile constituents. At a temperature of 275° F. it has been used for two years before getting too thick to circulate freely enough to keep the temperature of the whole quantity uniform, and avoid overheating a part of it. The usual life is 9 to 12 months. When too thick for further use, it makes an excellent rope compound. It preserves a rubbery consistency without hardening in cold weather, and does not melt. In fact, it has to be thinned somewhat before the rope takes it well.

Air Stored in Six Cylinders

An air-storage system, composed of six cylinders of 2,500 cu. ft. capacity each, is connected by a 24-in. pipe to a reservoir above. The reservoir holds 4 ft. of water, and is set at such a height that it exerts 93 to 91 lb. pressure on the tanks as it is full or empty. A drop in pressure from 93 lb. to 91 lb. will empty the reservoir and fill the air tanks with water, forcing 15,000 cu. ft. of air at 91 lb. or approximately 100,000 cu. ft. of free air back into the system. This eases the variable hoisting engine load.

Electric Hoist at Czar Shaft

The Czar engine was an old steam hoist for single-deck cages, built by the Union Iron Works, in San Francisco, about 30 years ago. The gear shaft was lifted out of its bearings, and replaced by another, geared to a 112-h.p. alternating-current motor. It has since been replaced by an electric hoist driven by a 225-h.p. alternating-current motor. It hoists a double-deck cage at 800 ft. per minute, but it is usually operated in

balance. It is driven through a direct line from the power house, where there is sufficient power in reserve to obviate the necessity of a fly-wheel set or other method of cutting down the peak load. It is started, however, by a General Electric automatic magnetic controller, to prevent too heavy a drag at the time of acceleration.

Brakes are operated by hand, the clutches by air pressure. Power for the latter was added after it was found that the method of throwing them by hand was too slow, when making many changes, as in cage hoisting it is necessary to do.

Several of the shafts are most inconveniently situated for waste disposal. At the Czar, Holbrook, and Lowell, the waste is dumped from the mine cars at the shaft into large cars holding 75 to 100 cu. ft., which are pulled out to the waste pile, by small hoists, and dumped automatically. One of them is pulled up an incline and returns by gravity, and one runs away by gravity and the empty car is pulled back by an engine.

Careful tests of the efficiency of different methods of hoisting were made, the conditions governing the tests being shown in the following paragraphs. Comparative results of the tests are shown in Table IV.

Conditions of Hoisting Tests

Test No. 1

Double-drum engine, reels 7 ft. diameter by 5 ft. face, $1\frac{1}{4}$ -in. rope. Driven by duplex, tandem compound, condensing steam engine, with Corliss valves.

Dimensions, 18 and 28 in. by 48 in. Cut-off by governor. Brakes, etc., actuated by hydraulic pressure, condenser and circulation pump by electricity

Hoisting $3\frac{1}{2}$ -ton skips in balance.

Average depth of hoisting 825 ft.; maximum depth 1,580 ft.

Boilers isolated for test and feed water weighed.

Steam main 5 in. by 619 ft.

Condensation in main 625 lb. per hour.

Ore weighed on conveyor at shaft, on Blake-Dennison scales.

Work done in changing levels calculated from weight of skip.

Test No. 2

Double-reel flat-rope engine, ropes $\frac{1}{2}$ by $5\frac{1}{2}$ in.

Driven by a 20 by 60 in. simple, duplex, non-condensing engine, with Corliss valves. Cut-off by governor. Brakes, etc. actuated by steam. Boilers close to hoisting engine.

Hoisting triple-deck cages in balance from a maximum depth of 1,600 ft.

Boilers isolated for test and feed water measured.

Ore or waste weighed on scales at collar of shaft.

Work done in changing levels calculated from weight of cages, etc.

Steam for auxiliaries was generated in another boiler, and amounted to 16.1 lb. per shaft horsepower hour, making a total consumption of 111.6 lb.

Test No. 3

Double-reel flat-rope engine, ropes $\frac{1}{2}$ by $5\frac{1}{2}$ in.

Same engine as No. 2, and same hoisting conditions, but engine driven by compressed air.

Air measured by displacement of compressor run especially for this engine.

Allowance is made for volumetric efficiency. Transmission losses are practically all leakage, not included, since the proportion chargeable to hoisting depends on the quantity of air passing through the main.

Air heated by coke heater.

Auxiliaries for brakes, etc., driven by cold air.

Ore or waste weighed on scales at collar of shaft.

Test No. 4

Double-reel flat-rope engine, ropes $\frac{1}{2}$ by $5\frac{1}{2}$ in.

Driven by a 21 by 60 in. simple, duplex, compressed-air engine, with Corliss valves. Cut-off by governor.

Hoisting triple-deck cages in balance from a maximum depth of 1,032 ft.

Air measured by displacement of compressor run especially for this engine, allowance being made for volumetric efficiency and line leakage, which was measured.

Air heated by coke heater.

Auxiliaries for brakes, etc., driven by cold air.

Ore or waste weighed on scales at collar of shaft.

Test No. 5

Double-reel flat-rope engine, ropes $\frac{3}{8}$ by $5\frac{1}{2}$ in.

Engine driven by a 20 by 48 in. simple, duplex, compressed-air engine, with Corliss valves, with cut-off by governor.

Hoisting triple-deck cages in balance from a maximum depth of 943 ft.

Air measured by displacement of compressor run especially for this engine, allowance being made for volumetric efficiency and line leakage, which was measured.

Air heated by coke heater.

Auxiliaries for brakes, etc., driven by cold air.

Ore or waste weighed on scales at collar of shaft.

Test No. 6

Double-reel flat-rope engine, ropes $\frac{3}{8}$ by 4 in.

Engine driven by a 16 by 42 in. simple, duplex, compressed-air engine with Corliss valves; cut-off by governor.

Hoisting double-deck cages in balance from a maximum depth of hoisting 645 ft.

Air measured by displacement of compressor run especially for this engine, allowance being made for volumetric efficiency and line leakage, which was measured.

Air heated by coke heater.

Auxiliaries for brakes, etc., driven by cold air.

Ore or waste weighed on scales at collar of shaft.

Test No. 7

Double-drum geared engine driven by 225-h.p. induction motor. Current three phase, 2,200 v.

Started by automatic magnetic control.

Hoisting double-deck cages in balance from a maximum depth of 417 ft.

Power measured by wattmeters at power house, and includes line losses.

Every fifth car of ore or waste was weighed on scales at the collar of the shaft. Equivalent pounds of steam and cost of power at power house per shaft horsepower hour are high, due to inefficient turbines at power house and also low load factor.

Test No. 8

Conditions similar to Test No. 7.

TABLE IV.—*Tests on Hoisting Efficiency*

Test Number	1	2	3	4	5	6	7	8
Date of test.....	4-19-11	4-6-09	2-18-12	2-25-12	3-3-12	3-3-12	6-4-15	6-14-15
Duration of test.....	8 hr.	7½ hr.	2 hr. 4 min.	1 hr. 16 min.	49 min.	30 min.	6 hr. 30 min.	3 hr. 15 min.
Vacuum, inches.....	19.8							
Motive power.....	Steam	Steam	Air	Air	Air	Air	Electric- ity	Electric- ity
Pressure gauge, pounds	137	125	89	89	87	88		
Temperature, ° F.....			205	150	220	200		
Work done, shaft horsepower hours.....	897.46	335.20	82.5	45.94	22.13	4.75	48.92	33.23
Power consumed per shaft horsepower hour, hoist alone.....	29.08 lb. steam	95.5 lb. steam	1,489 cu. ft. free air	1,966 cu. ft. free air	1,897 cu. ft. free air	2,322 cu. ft. free air		2.47 kw-hr.
Including condenser.....	31.94 lb. steam							
Including line loss.....	34.29 lb. steam						2.93 kw-hr.	2.6 kw-hr.
Including auxiliaries.....	34.29 lb. steam	111.6 lb. steam						
Including reheating.....			1,602 cu. ft. free air	2,177 cu. ft. free air	2,037 cu. ft. free air	2,494 cu. ft. free air		
Equivalent pounds steam per shaft horsepower hour at power house.....	34.29	105.07	52.87	71.84	67.22	82.30	82.04	72.8
Cost of power per shaft horsepower hour.....	\$0.0137	\$0.0399	\$0.0288	\$0.0392	\$0.0366	\$0.0448	\$0.0465	\$0.0416

In Test No. 2, the 111.6 lb. of saturated steam at 125 lb. pressure is reduced to 105.07 lb. in order to make an equitable comparison in efficiency of all tests. In all other tests, the equivalent pounds of steam are based on 150-lb. pressure with 75° F. of superheat, which are the power-house conditions.

In Tests Nos. 3, 4, 5, and 6, the cost of re-heating air appears as an amount of compressed air equal to it in cost.

In Test No. 7, the estimated line loss is 3 per cent.

It is assumed from power-house figures that 33 lb. of steam will compress 1,000 cu. ft. of free air to 93 lb. gauge.

It is also assumed that 28 lb. of steam at the power house generates 1 kw-hr. at the power-house switchboard.

The cost of available steam at power-house is 45 c. per 1,000 lb.

The cost of air at the power-house is 1.8 c. per 1,000 cu. ft. of free air compressed to 93 lb. gauge.

The cost of electricity at the power house is 1.59 c. per effective kilowatt-hour at the power-house switchboard.

The double- and triple-deck cage hoists are at a disadvantage compared to skip hoisting, since two or three lifts of 6 to 8 ft. must be made to land the decks. This only appears as so many feet of hoisting in work done, but it means getting the load in motion, which is the least efficient part of the work, twice or thrice instead of once.

This is exaggerated in the case of the Czar hoist by the shallow depth of hoisting. It will be noted that the decrease in power per shaft horsepower hour in test No. 8 as compared to No. 7 is due to a greater percentage of the material to be hoisted coming from the 400 level. When hoisting from that level, the current consumption per shaft horsepower hour was 2.29 kw.-hr.; from the 200 level it was 2.92 and from the 100 level 4.6 kw.-hr. per shaft horsepower hour. The record of this test is not representative of electric hoisting, and is included as a matter of interest only. Another hoist of the same type will shortly be erected at a 900-ft. shaft, and further tests will be made.

The simple steam hoist is inefficient at best, and particularly so in intermittent service, which does not, however, particularly affect either the compressed-air hoist or an electric hoist without fly-wheel motor generator.

It was a great economy to be able to use the hoisting engines as they stood, even if they were not quite so efficient as those especially designed for air.

TABLE V.—*Ore Haulage by Locomotive*

	1914, Cents per Ton	Four Months, 1915, Cents per Ton
Train labor.....	6.0	3.5
Loading labor.....	4.0	4.5
Motor maintenance.....	2.0	1.1
Car maintenance.....	2.2	1.3
Trolley maintenance.....	1.2	0.6
Track maintenance.....	2.0	1.4
Transfer chutes.....	2.2	2.8
Power.....	1.3	1.3
Miscellaneous.....	1.1	2.2
Foremen and bosses.....	1.0	0.9
	23.0	19.6

The average distance trammed per car in 1914 was 2,939 ft., returning empty.

Car repairs are rather higher than normal, since the equipment is growing old. Most of the cars have been in service for five years.

The charge for labor indicates that one man takes care of three to four miles of main trolley line, as well as bonding, and a proportion of the transmission line in the shaft. Trolley lines in heavy ground require much attention.

Track maintenance includes renewals and all other repairs. One man cares for $1\frac{1}{2}$ miles of track. Track and trolley extensions are not included. In 1914, this amounted to 5.3 c. per ton.

In 1914, 1.2 c. per ton were spent in transfer-chute construction, and an equal amount in 1915. Much permanent work has been done, and future costs are expected to be lower.

TABLE VI.—*Ore Trimming by Hand or Mule*

	1914	Four Months, 1915
Labor.....	10.8	9.5
Mules.....	1.0	1.3
Car repairs.....	1.6	1.3
Track repairs.....	3.2	3.3
Miscellaneous.....	3.3	2.8
Foremen and bosses.....	0.8	0.8
	20.7	19.0

The cost of hand and mule trimming is so intermingled that it is impossible to separate them. Ore is trimmed 194 ft. by hand and 708 ft. by mule. All new light track is charged to this account, whether for new drifts or replacements. In 1914, 47,800 ft. of drift were driven.

TABLE VII.—*Ore Hoisting, Sacramento Shaft*

	1914		Four Months, 1915	
	Total Cost	Cost per Ton	Total Cost	Cost per Ton
<i>Power:</i>				
Steam.....	\$12,632.12	\$0.019	\$4,216.43	\$0.023
Electric.....	436.72	0.001		
<i>Attendance:</i>				
Engineers.....	5,694.54	0.009	1,848.25	0.010
Oilers, etc.....	3,017.44	0.005	931.92	0.005
Skip loaders.....	7,585.19	0.012	1,844.18	0.010
Cleaning stations.....	214.30			
Miscellaneous.....	495.48	0.001	800.26	0.004
Hoist repairs.....	8,702.82	0.013	1,093.63	0.006
Cage repairs.....	2,549.03	0.004	625.60	0.003
Ore pockets.....	1,537.01	0.002	981.79	0.005
Hoist buildings.....	9.05		1.51	
Ropes.....	3,252.43	0.005	586.39	0.003
Total.....	46,126.13	0.071	12,929.96	0.069
Tons hoisted.....	659,102		185,179	

The cost of hoisting is divided into two classes, that of ore hoisting at the Sacramento shaft, and that of handling men, timbers, and waste rock at that and other shafts.

TABLE VIII.—*Conveyor-Plant Costs*

	1914	Cost per Ton Four Months, 1915
Power.....	\$0.0037	\$0.0027
Attendance.....	0.0122	0.0129
Belts.....	0.0026	0.0068
Other supplies.....	0.0007	0.0008
Repairs.....	0.0097	0.0078
	0.0289	0.0310
Cost of screening.....	0.0015	0.0021
	0.0274	0.0289

The rope cost includes that of some defective ropes on which a credit is expected.

Belts are usually charged off more rapidly than they wear out. A better idea of the cost is given by the records of the last belts on each conveyor.

TABLE IX.—*Conveyor-Belt Records*

Belt	A	B	C	D	Tripper
Width, inches.....	30	30	30	30	36
Length, feet.....	415	700	680	60	16½
Tonnage handled.....	1,166,734	1,070,363	984,130	353,621	51,710
Cost per ton, cents.....	0.122	0.242	0.288	0.087	0.226
Cost per ton all belts, cents.....				0.965	

Cage riders were not customarily employed before the change in the hoisting arrangement. They were found to reduce the danger of shaft accidents.

It will be noted in Table X that the costs at the shafts handling on ore are much more in the aggregate than those at the Sacramento shaft. They are necessary evils, to be held down to the lowest terms.

In 1907, the usual equipment for a producing shaft was: First-motion double-reel engine; double- or triple-deck cages; single-drum man engine; single-deck cage; boiler plant; blacksmith shop; drill- and tool-sharpening shop; timber-framing mill; drill repair shop.

In 1915, the plant is reduced to the first-motion hoist, to which an air heater has been added. The drill- and tool-sharpening shop, framing mill and drill repair shop were not eliminated by the change in hoisting methods, but followed the tendency toward centralization. The ex-

pense for surface labor has been reduced in proportion, although it appears principally as a credit to mechanical labor and other surface accounts. The fuel consumption has been approximately cut in two, although the change rooms at each shaft must be heated by independent plants instead of by exhaust steam from the engines.

TABLE X.—*Cost of Handling Men, Timber and Waste Rock*

	Uncle Sam	South- west	Csar	Hol- brook	Spray	Gard- ner	Lowell	Sacra- mento Man Engine	Total	Per Ton
Power:										
Steam.....								\$7,634	\$7,634	\$0.011
Air.....	\$1,100			\$2,667	\$628	\$6,540	\$6,023		16,958	0.026
Air reheaters.....				565	158	911	492		2,125	0.003
Electric.....		\$744	\$1,472	71		747	234	243	3,512	0.004
Labor:										
Engineers.....	885	4,816	3,799	5,724	1,387	7,358	5,004	4,808	33,779	0.049
Oilers.....				1,662	548	1,782	1,997	1,666	7,655	0.011
Cagers.....	739	2,404	4,349	6,071	808	9,233	5,421	2,837	31,861	0.047
Cleaning stations.....			329	133	115	376	687		1,642	0.002
Miscellaneous.....				105					105	
Hoist repairs.....	886	1,036	2,416	1,475	519	2,283	2,550	1,695	12,860	0.020
Cage repairs.....	12	250	2,035	850	292	756	1,641	706	6,552	0.010
Rope repairs.....	90	93							183	
Ore pockets.....			116						116	
Hoist buildings.....			511						511	0.001
Rope maintenance.....			32	315	1	1,701	927	113	3,272	0.005
	3,712	9,352	5,059	19,638	4,456	31,687	24,976	19,702	128,582	0.189
Four Months, 1915										
Power:										
Steam.....								1,985	1,985	0.010
Air.....				858		1,927	1,626		4,411	0.022
Air reheaters.....				327		229	245		800	0.004
Electric.....		380	619	18		312	1		1,329	0.007
Labor:										
Engineers.....		766	1,216	1,848		2,209	1,266	1,172	8,478	0.042
Oilers.....				544		541	654	546	2,284	0.012
Cagers.....		640	1,600	1,621		2,036	1,216	992	8,106	0.040
Cleaning stations.....			176	116		43	87		422	0.002
Miscellaneous.....				89					89	
Hoist repairs.....		283	427	442		466	699	77	2,394	0.010
Cage repairs.....		53	345	192		855	1,283	457	3,185	0.015
Hoist buildings.....		23	10				27		60	
Rope maintenance.....		165	104	141		626	613	3	1,653	0.009
	2,310	4,497	6,196			9,244	7,717	5,232	35,196	0.173

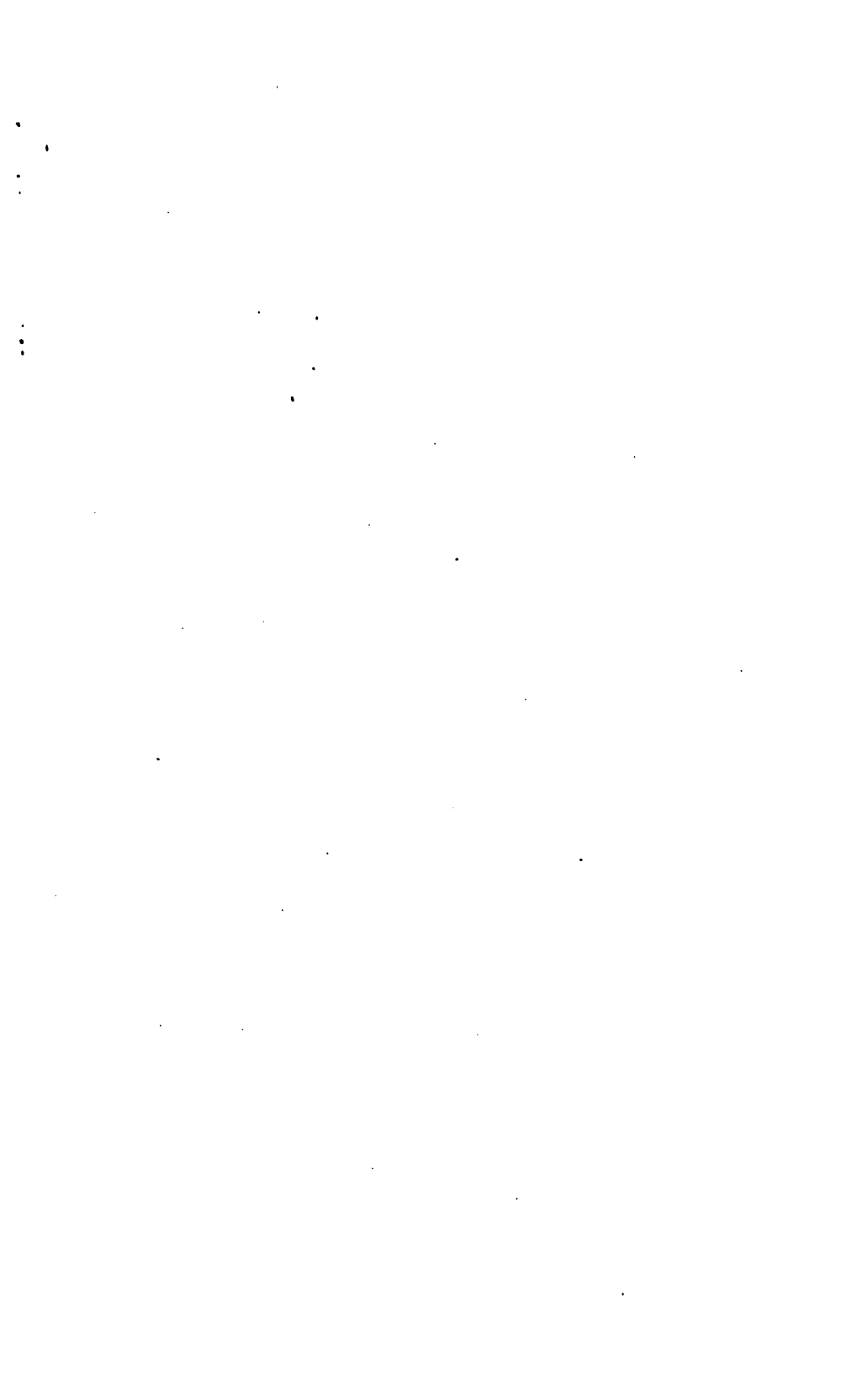
As a whole, the costs of tramming and hoisting have not been reduced as much below the costs of 1906 as expected. A reduction in costs due to an improvement in method has been counteracted by the gradually

increasing area of operation and necessary equipment. If a comparison might be made with former methods, developed and extended to cover the present work, it would be all that was claimed for it and more.

Although a saving in repairs was counted upon in the original estimates, it was not specified. In 1914, underground repairs cost 51.2 c. per ton and in the first four months of 1915, 40 c. During the last three months of 1905, they cost 59.8 c. for labor alone, and for the first six months of 1906 the cost was 62 c. for labor and supplies.

Individual shafts are not so vitally essential as before. In 1911, the gases from a mine fire rendered the Lowell shaft impassable to men between the 800 level and the surface, for about five months. For two months, timber and supplies were lowered and waste was hoisted through the gas, but since no repairs could be made, a guide finally broke loose and jammed one of the cages. During this period, workmen were lowered through neighboring shafts and ore was hoisted as usual. Both production and costs for the year were normal.

The writer wishes to express his indebtedness to members of the mine department staff for assistance in collecting data used in this paper; W. Saben, Chief Clerk, Charles Legrand, Consulting Mechanical Engineer, R. E. Cameron, a member of his staff, and George Mieyr, Master Mechanic.



TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

The Pacific Coast Iron Situation

The Iron Ores of California and Possibilities of Smelting

BY CHARLES COLCOCK JONES,* B. S., LOS ANGELES, CAL.

(San Francisco Meeting, September, 1915)

IN any discussion of this very large subject we are confronted at the outset with so many obstacles that at best only a fragmentary and rather disconnected presentation can be made of it, and my hope is that building from the few facts I have to offer a more complete knowledge may be secured.

In the whole intermountain and Pacific Coast region, west of Pueblo, Col., and from Canada to Mexico, there is no production of pig iron, and steel production is confined to the manufacture of a small amount of open-hearth steel from scrap. When we consider the area, the resources, and the several centers of population, this condition seems anomalous, even in the face of every counter argument as to cheap water transportation, or other means of transport from the Atlantic Seaboard or from the iron and steel centers of the Middle West.

This comatose condition, if such a term may be applied, is certainly not due to a lack of all the essential materials and factors for success in iron and steel smelting, but is rather, I think, due largely to the general fact that it is only within a comparatively few years that the centers of population on the Pacific Coast have attained a size that would entitle them to the position of distributing or freight centers, or that the "back country" has been enough built up to make it sufficiently interesting to the railroads to foster trade from the local centers to the interior points rather than seek to supply such local centers and the interior by a transcontinental haul.

ORE SUPPLY

Mr. Edwin C. Eckel, in his monumental work on *Iron Ores*, has but little to say concerning the Pacific Coast deposits. He mentions (p. 269) those of Minaret, Madua County, and the Eagle Mountains, Riverside County, Cal., "perhaps 50 to 100 million tons each;" the deposits in Shasta County, worked for the electric furnace on Pitt River; and

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"ores in Oregon and Washington" which "have been slightly developed;" and while he admits that (by reason of unfavorable conditions, as to labor and fuel, for the manufacture of iron) there has been comparatively little prospecting for iron ores in the Pacific States, he thinks that certain geological types of iron-ore deposits, which have been highly productive in the East, are probably wanting here—for instance, sedimentary beds like the Clinton, and brown-ore deposits like those of the Appalachian region. And in his final summary of American iron-ore resources (p. 349) he cites Hayes' estimate of 126,000,000 tons of available ore "for the Rocky Mountain and Pacific States;" pronounces it too low, since Utah alone contains more; and finally puts down the Western States for a minimum of 300,000,000 and a possible maximum of 700,000,000 tons. His estimate for the whole United States being 5,200,000,000 minimum and 7,500,000,000 maximum, the proportion assigned to the great area between the Rocky Mountains and the Pacific is notably small.

Mr. Eckel does not cite, in his Bibliography of California deposits, *Bulletin 503, U. S. Geological Survey*, by E. C. Harder, on Iron Deposits of the Eagle Mountains, published in 1912. In the light of the Harder report on the Eagle Mountains and other data, I think the Eckel estimate of from 300,000,000 to 700,000,000 tons can be safely applied to the Pacific States alone, excluding the tonnages of Utah and the other Rocky Mountain States; and, from our present knowledge, most of it is in Southern California.

At the annual meeting of the American Mining Congress held in Los Angeles in 1910, the writer, in an address on the subject of Iron Ores of the Southwest, called attention to the large available deposits of high-grade iron ore in southern California and adjacent territory, and previously in the *Engineering and Mining Journal* of April 17, 1909, had described technically the iron deposits in the Providence Mountains of San Bernardino County, California. The U. S. Geological Survey published in 1908, *Bulletin 338 on The Iron Ores of the Iron Springs District, Southern Utah*," and in 1912, *Bulletin 503, Iron Deposits of the Eagle Mountains, California*. Descriptions of various other iron deposits are scattered through technical publications.

Epitomizing the subject in 1910, I came to the conclusion that there are 200,000,000 tons of available, high-grade iron ore and double that amount of probable ore in southern California. In addition to this an equal amount can be counted on from Lower California and Mexico, which will naturally seek an outlet to the northward.

This estimate does not include the Utah deposits.

In 1908 and 1909, the late Edward H. Harriman acquired the Eagle Mountain iron deposits, in Riverside County, 50 miles north of Mecca, a station on the Southern Pacific Railroad, 140 miles east of Los Angeles,

with the publicly avowed intention of establishing a steel works at Los Angeles. The U. S. Geological Survey states as a "moderate estimate" that there are 60,000,000 tons of iron ore in this deposit. Later, what is known as the Cave Canyon iron deposit, 200 miles east of Los Angeles on the Salt Lake Railway, containing 20,000,000 tons, was bought by the same interests. The untimely death of Mr. Harriman in 1909 put a damper on further development, and his vision is yet to be realized.

Near Kelso, 236 miles east of Los Angeles, is the Vulcan iron deposit, containing from 6,000,000 to 10,000,000 tons of high-grade, soft hematite ore of Bessemer grade.

In the Kingston Mountains at the California-Nevada line, north of the Salt Lake Railway are other large deposits.

The Colorado Fuel & Iron Co. owns a deposit of 12,000,000 tons 10 miles west of Silver Lake on the Tonopah and Tidewater Railway.

There are large amounts on the Santa Fe Railway in San Bernardino County south of Newberry, north of Amboy and south of Blythe Junction, and numerous other deposits are known in the desert region. The three most practically available deposits, viz.: the Eagle Mountain, Cave Canyon, and Providence Mountain deposits, represent an aggregate tonnage of 125,000,000 tons, and have the characteristics necessary to the best possible furnace mixture for economical smelting. These ores run from 4 to 10 per cent. higher than standard Lake Superior iron ores. The bulk of the tonnage is within the Bessemer limit in phosphorus and, lacking moisture, there is no cost of drying before transportation, or freight cost on 10 per cent. of moisture as with Lake ores. *Bulletin 503*, on Iron-Ore Deposits of Eagle Mountain, California is authority for these statements. The average analysis of Providence Mountain iron ore is:

Iron, 64.82; phosphorus, 0.044; silica, 3.04; manganese, 0.278; alumina, 0.568; magnesia, 0.201; lime, 0.444; sulphur, 0.059; titanium, none; volatile, 1.85 per cent.

CONSUMPTION

According to one of the leading buyers of iron and steel, the following is a conservative estimate of the present approximate yearly consumption on the Pacific Coast:

	Tons
Pig iron.....	100,000
Reinforcing steel for concrete buildings.....	30,000
Sheets.....	136,000
Plates.....	217,000
Corrugated.....	15,000
Bar.....	175,000
Structural.....	250,000
	923,000

Probably the aggregate is more nearly double this estimate. Complete figures would include the large amounts of cast-iron water pipe, oil-well pipe, railroad steel and other steel products not listed above.

COSTS OF PRODUCTION

At some future time there will certainly be one iron-smelting center in Southern California; and there may possibly be a second at San Francisco and a third at Seattle, each having its own distinct province from which to draw its supplies of raw materials; but at the present time Southern California seems to have the advantage in the combination of ore, fuel, labor and climatic conditions for economical production.

The statement that conditions justify the establishment of at least one plant, based on the consumption of iron and steel products on the Pacific Coast will be borne out, if it can be shown that these products can be manufactured locally at a price to compete with the supply from other smelting centers.

In the following discussion some of the figures relating to Pittsburgh are used as given in a recent publication by the Merchants' Association of New York on the iron and steel industry, supplemented by personal knowledge, and figures from *Mineral Resources of the United States*, 1913, published by the U. S. Geological Survey, and also figures from the report of the Commissioner of Corporations on the Steel Industry, 1911, and costs of making pig iron from testimony before the Ways and Means Committee of the House of Representatives in 1906.

Pittsburgh is the center of the iron and steel industry in the United States and any costs on the Pacific Coast that can be shown to approximate Pittsburgh costs will demonstrate not only the feasibility of the industry, but the certainty of profits.

The difference in mining and transportation costs between Lake Superior ores and California ores is due to a number of reasons, principal among which are the following: The Lake Superior ores are obtained either by expensive underground mining or by open-cut work in which large amounts of overburden have to be removed at considerable cost; and, on account of climatic conditions, the shipping season is limited to seven months of the year. Transportation is a combination of rail, water and rail, entailing several handlings of the ore.

In the case of California ores, we have large surface exposures of orebodies, capable of being quarried and loaded on cars by steam shovel, and handled thence direct to the furnaces. The mining cost of California iron ores will be on the same low basis as that of the porphyry copper ores, or from 20 to 30c. per ton. The figure of 85c. in the following estimate is ample to cover both royalty, or sinking fund, and mining. Climatic conditions are such that work can be carried on every day in

the year with the exception possibly of a few stormy days; and California ores have the added advantage of a higher content in iron and a lower amount of gangue to be fluxed, resulting in greater efficiency, or tonnage product of a furnace and decreased cost of flux, per ton of iron produced, all of which factors materially lower the operating cost per ton of pig iron produced.

TABLE I.—*Comparison of Costs of Iron Ore and Transportation at Pittsburgh and Los Angeles*

	Pittsburgh Lake Superior Ores	Los Angeles Cali- fornia Ores. As- suming Max. Haul 280 Miles at 0.005c.
Rail and Lake rate to Lower Lake Ports.....	\$1.15	
Rail freight rate, Lower Lake Ports to Furnace location.	0.88	
Rail freight rate, Eagle Mountains, Providence Mountains, and San Bernardino County ores.		\$1.30
Total transportation, costs per ton.....	\$2.03	\$1.30
Average mining cost per ton.....	\$2.40	\$0.85
Total transportation and mining cost per ton.....	\$4.43	\$2.15
Percentage iron yield per ton ore, non-Bessemer grade, per cent.	51.5	57.00
Percentage iron yield per ton ore, Bessemer grade, per cent.	55.0	64.00
Total transportation and mining cost per ton pig iron, 95 units iron, non-Bessemer grade.	\$8.17	\$3.56
Total transportation and mining cost per ton pig iron, Bessemer grade.	\$8.05	\$3.19

Comparing the Bessemer ores alone, as the bulk of the California ore is of that quality, the advantage of Los Angeles over Pittsburgh, in cost of iron ore per ton of pig iron is \$4.86.

The average mining cost for Lake Superior ores is taken from *Mineral Resources of the United States*, 1913, U. S. Geological Survey, and is exclusive of any transportation costs.

Coke Supply

Of equal importance to the iron ore is a supply of suitable coal for coking. There are probably good coking coals in Alaska and Washington that could be placed in the Los Angeles market at a reasonable price; and there are large coal fields in southern Utah at present without railway connection, undoubtedly containing coking coals; but at the present time the coking coal of Carbon County, Utah, is the most available by reason of rail connection. The Carbon County, Utah, coal, at Sunnyside in the Book Cliffs coal field, compares favorably in coking and

chemical qualities with some of the Eastern coals and a comparison of costs on that basis will be made.

An average analysis of Sunnyside coal and coke is as follows:

	Coal, Per Cent.	Coke, Per Cent.
Moisture.....	1.27	0.13
Volatile matter.....	39.13	0.83
Fixed carbon.....	54.62	88.96
Ash.....	4.98	10.07
	100.00	99.99

The metallurgical coke for the Utah and Montana copper smelters is supplied from the Sunnyside field and its good qualities are thoroughly proven. A description of the Book Cliffs Coal Field was given by Robert S. Lewis.¹

In the report of the Commissioner of Corporations on Steel Industry, 1911, the cost of coke at Pittsburgh has been shown to be \$2.97 per ton of pig iron, and from that cost it is estimated that the cost of byproduct coke, provided a profitable market for the byproducts could be secured at Pittsburgh, would be \$1.94 per ton of pig iron. The above figure of \$2.97 is borne out by the current quotations on Connellsville coke, May 1, 1915. As compared with the cost of byproduct coke at Pittsburgh, the following costs are conservative for Los Angeles, where a ready market could be found for the byproducts and where a number of chemical industries could follow the establishment of a byproduct coke plant, utilizing the coal tar, ammonia and gas.

TABLE II.—*Cost of Coke at Los Angeles or Southern California Points*

Mining cost 1.65 tons Utah coal at 90c. per ton.....		\$1.48
Transportation 1.65 tons at \$4.25, 850 miles at 0.5c.....		7.01
Total cost coking and byproduct operation per ton coke.....		1.40
		9.89
Credit from sale of byproducts per ton coke, 8,000 cu. ft. gas at 15c... 14 gal. tar at 4c.....	\$1.20	0.56
24 lb. sulphate ammonia at 3½c.....	0.84	2.60
Net cost of coke per ton.....		\$7.29
Total cost of coke per ton of pig iron (1.10 tons per ton pig).....		\$8.02
Cost byproduct coke Pittsburgh, minimum.....		1.94
		6.08
Maximum advantage of Pittsburgh in cost of coke per ton of pig iron.		
Total cost of coke per ton of pig iron (1.10 tons per ton pig).....	8.02	
Or at average cost of coke.....	2.97	
Average advantage of Pittsburgh in cost of coke per ton of pig iron...	\$5.05	

¹ *Bulletin* No. 91, July, 1914, pp. 1729 to 1749.

The building of the Alaska Railroad into the Matanuska coal field by the government will open that coal to Southern California; and, if the claims regarding its coking qualities are substantiated, it will afford a supply of coking coal cheaper than from any other section.

In Table II the coking question has been treated from the standpoint of making byproduct coke at tidewater in order to improve the coke, secure the byproducts, and have a source of supply of power in the waste gases for use in a steel plant independent of any one coal field; but the coke supply can also be considered from the standpoint of beehive-oven coke made in the coal fields and transported to tidewater, if the cost of such coke can be shown to be less than that of byproduct coke.

I should observe that my estimates are made on a rational freight rate of 0.5c. a ton a mile. The present published rates on coal and coke from the interior are higher than that, being still based on "all the traffic will bear;" and, since the amount handled is small, no attempt has been made to secure a lower rate. That 0.5c. a ton a mile is a reasonable maximum is shown by the fact that that rate is in existence for phosphate rock and other material, of a like or higher value than coal, from this intermountain region to tidewater. I mention this matter because it is one of the "scarecrow arguments" that have to be met. When the traffic justifies it, the rate can be secured, through the usual channels.

Limestone Supply

Available to every railroad entering Los Angeles, at distances of from 20 to 150 miles, are ample deposits of limestone suitable for blast-furnace flux. This rock can be quarried and brought to tidewater at as low a price as that used in Pittsburgh, and from many of the deposits for a less cost. Even at a greater cost per ton of limestone, the cost per ton of pig iron would still be as low as the Pittsburgh cost on account of the greater purity of the iron ores and consequent smaller amount of lime used for flux. In my estimate of final costs, the Pittsburgh figure will be applied as fully covering the limestone cost.

Application of Above Figures and Comparison of Cost of Pig Iron at Pittsburgh with the Estimated Cost at Los Angeles

In 1906, before the Ways and Means Committee of the House of Representatives, Judge Gary testified that for all the furnaces of the United States Steel Corporation the total cost of making a ton of pig iron, based on the market prices of iron ores, coke, limestone, cinder and scale, and including maintenance and depreciation of furnaces, was \$15.30 per ton of pig iron, divided as follows:

TABLE III.—*Cost of Pig Iron at Pittsburgh*

Iron ore per ton \$4.70, per ton pig iron.....	\$8.62
Coke per ton \$3.93, per ton pig iron.....	4.15
Limestone \$1.06, per ton pig iron.....	0.49
Scrap, cinder and scale.....	0.27
Labor and maintenance.....	1.37
Depreciation of furnaces.....	0.40
	\$15.30

From these figures he said the following deductions should be made for net profits:

On ores in pig iron.....	\$2.04	
On coke in pig iron.....	0.60	
On transportation.....	1.07	\$3.71
Actual cost of pig iron.....		\$11.59

Assuming other costs than the ore, coke and flux to be the same at Los Angeles and Pittsburgh, we have the following as a conservative statement of the estimated cost of a ton of pig iron at Los Angeles, or tidewater.

TABLE IV.—*Estimated Cost of Pig Iron at Los Angeles*

From Table I—Total transportation and mining cost per ton pig iron, Bessemer grade ore.....	\$3.19
From Table II—Total cost of coke per ton pig iron.....	8.02
Limestone.....	0.49
Scrap, cinder and scale.....	0.27
Labor and maintenance at furnaces.....	1.37
Depreciation of furnaces.....	0.40
	\$13.74

In <i>Mineral Resources</i> , 1913, Part 1, p. 321, the government gives as the average price per ton of pig iron at the Pennsylvania blast furnaces for 1913.....	\$15.36
And for Alabama furnaces.....	12.08

Accepting for the purposes of this discussion the figures given by Mr. Gary of \$11.59 as the net cost of pig iron for all the furnaces of the United States Steel Corporation in the year 1906, and deducting it from the estimated cost at Los Angeles of \$13.74, we have a difference of \$2.15 per ton in favor of Pittsburgh, but the railroad freight rate on pig iron from Pittsburgh makes the net cost of pig iron at least \$20.95 f.o.b. Los Angeles and selling price \$23.00 to \$25.00 per ton.

In regard to freight rates on pig iron and steel products to Pacific Coast points, the published rate from Chicago and Birmingham is \$9 per ton, and there is no through rate from the Pittsburgh district, and from Pittsburgh to Chicago takes a local rate of \$3.15, making the total freight from that point \$12.15 per ton to California terminal points.

Steel products have a uniform freight rate of \$16 per ton, Pittsburgh to Coast points. In my calculations an average freight rate of \$9 has been used, being the rate from Chicago or Birmingham to Coast points.

It should be noted particularly that our comparison of costs is with the costs of a corporation owning mines, railroads, vessels and smelting plants and its costs are necessarily lower than the merchant-furnaces in Pennsylvania which buy ore and coke, and a deduction from Tables I and II more nearly states the relative costs.

From Table II, advantage of Pittsburgh in cost of coke, per ton of pig iron.....	\$6.08
From Table I, advantage of Los Angeles in cost of ore, per ton of pig iron.....	\$4.86
Net advantage of Pittsburgh in cost of ore and coke per ton of pig iron.....	\$1.22

If, however, in all fairness to Southern California, the figure \$5.05, the *average advantage*, instead of \$6.08, the *maximum advantage* of Pittsburgh in cost of coke per ton of pig iron is used, as shown in Table II, we have 19c. only as the *net advantage* of Pittsburgh in cost of ore and coke per ton of pig iron. *In other words, iron ore is the costly item and coke the cheap item at Pittsburgh, while in California the costly item is coke, and iron ore is the cheap item, with the difference in the cost of the two materials about equalized.*

From Table I, the cost per unit of iron, Lake Superior iron ore, Bessemer grade at Pittsburgh, is 8c.; the cost of California iron ore, per unit of iron, at Los Angeles is 3.33c.

Markets and the Cost of Reaching Them

At the present time the entire Pacific Coast is dependent upon the East and South for its supply of pig iron and steel, with the exception of a small amount of open-hearth steel produced from scrap at San Francisco and Los Angeles. The imports of foreign iron and steel are a negligible quantity and are never likely to be of any importance on account of the cost of transportation.

Therefore, as our principal market, we have this Pacific Coast province, comprising the States of California, Oregon and Washington, which can only be reached by the eastern manufacturer after paying a minimum transcontinental freight of \$9 per ton, or a minimum water freight from Atlantic ports of \$6 per ton.

Los Angeles is the great lumber market of the Pacific Coast, and most of the vessels in the trade are without cargo on the return trip and would welcome the opportunity to carry iron and steel products to San Francisco, Portland, Seattle and Vancouver at low water-rates. The markets of South America are open on more than equal terms; for tonnage through the Panama Canal from the Atlantic seaboard to the west coast of Mexico and Central and South America must pay the toll of \$1.20 per ton.

Simply stated, pig iron can be made in the regular blast furnaceway at Los Angeles for less than \$14. No Pennsylvania pig iron, either by transcontinental freight or by rail to New York and via Panama Canal, can be put into this market under \$19 per ton. Alabama pig might be put in at a slightly less cost, but its quality would not place it in the same class as pig iron made from California ores.

Geology

The principal Southern California iron deposits are of the contact-replacement type formed at or near the contact of an intrusive igneous mass with sedimentary strata, usually limestones. Their occurrence has been fully discussed in Iron Ores of the Iron Springs District, Southern Utah, by C. K. Leith and E. C. Harder, *Bulletin* 338, *U. S. Geological Survey*, 1908; Iron Ore Deposits of the Eagle Mountains, California, by E. C. Harder, *Bulletin* 503, *U. S. Geological Survey* (1912); and An Iron-ore Deposit in the California Desert Region, by C. C. Jones, *Engineering and Mining Journal* (April 17, 1909); and to these accounts I will add only the following note, bearing somewhat on a recent discussion in the Institute.

The Providence Mountains of Eastern San Bernardino County present the greatest uplift and cross-section of strata found between the Grand Canyon of the Colorado in Northern Arizona and the Sierra Madre range in Western San Bernardino County, and afford an opportunity to study a section approximately 4,000 ft. thick, from the underlying granitic and schistose rocks up through a sedimentary series predominantly composed of limestones with subordinate beds of shale and quartzite. This has been brought to the attention of the U. S. Geological Survey; and I believe, an extended study of the region will fill several gaps in our knowledge of the geology of this part of the desert region.

The general strike of the sedimentary beds is northeast-southwest with a slight dip to the east. The central core of the mountain, exposed in canyon from base to top, and forming the highest peaks, is a core or plug of rhyolite, a mile or more in diameter. At the southwest end of the mountain the sedimentary beds, lying horizontal or dipping slightly to the east, and retaining this position undisturbed, have been cut by a large quartz-monzonite mass, the contact between the monzonite and the limestones being practically vertical. I have indicated on the accompanying sketch map the general conditions.

The specific fact I wish to bring out for the attention of those interested in the discussion on The Origin of the Garnet Zones and Associated Ore Deposits,² by Waldemar Lindgren, and Recrystallization of Lime-

² Trans., xlviii, 201 to 208 (1914).

stone at Igneous Contacts,² by C. K. Leith, is the different alterations sustained by the same series of limestones in adjacent areas in contact with the same igneous intrusive rock mass.

In the sketch map, I have indicated conditions 9 miles southeast of Kelso. *R* is the central core of this part of the Providence Mountains, composed of rhyolite, and detached small rhyolite exposures are shown at several places in the limestones marked *R*. *XY* is the line of contact between the sedimentary series and the quartz-monzonite mass.

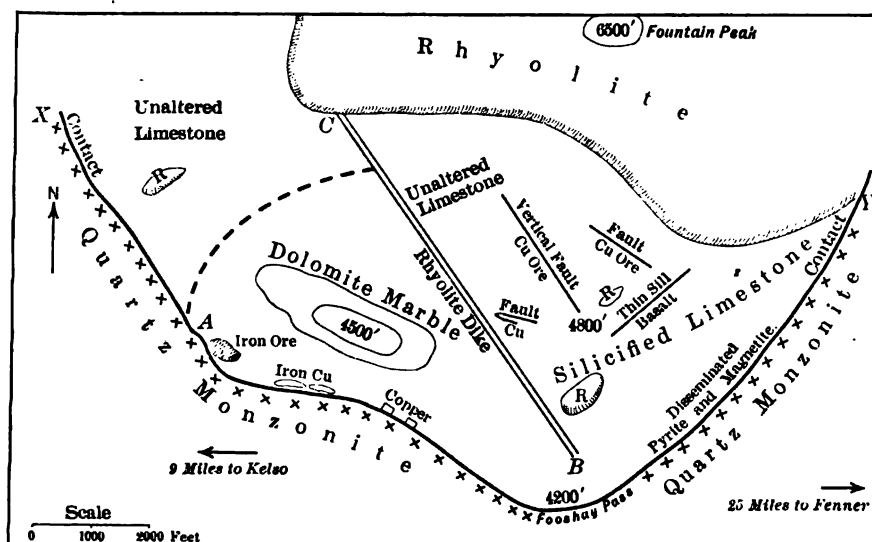


FIG. 1.—SKETCH MAP SHOWING GEOLOGICAL CONDITIONS IN EASTERN SAN BERNARDINO COUNTY, CAL.

The contact of the monzonite, cutting sharply the practically horizontal beds of limestone, runs for a distance of about 4 miles. *BC* is a vertical dike of rhyolite porphyry, 100 ft. thick, an offshoot from the central core. The point marked *A* seems to have been the center of activity, resulting in a replacement of the limestone by iron, over an area of four acres, with smaller orebodies along the contact to the east and with a great dissemination of grains and irregular replacements of iron ore in the limestones. The original bedding planes of the limestone are still preserved in the beds of iron ore; and in the area *ABC* the limestones have largely lost their bedded structure and a mass of crystalline dolomite or dolomitic marble has resulted. A fuller description of the geologic features is given in my article in the *Engineering and Mining Journal* cited above. The end of the dike *BC* is not seen in actual

² Trans., xlviii, 209 to 215 (1914).

contact with the monzonite at *B*. To the west of dike *BC* is an area of dolomite, as indicated, of the following analysis:

	Per Cent.
Insoluble siliceous.....	0.43
Aluminum and iron oxides.....	0.24
Calcium carbonate.....	63.08
Magnesium carbonate.....	35.97
	99.72

To the east of dike *BC* is an area of limestones between the rhyolite and monzonite, a portion of which is seemingly unaltered, especially just east of point *C*, where, in contrast with the dolomite west of the dike, the blue unaltered limestones preserve perfectly their bedded structure, and the larger portion of the area, especially at and for several hundred feet north of the contact, is largely silicified. An analysis of a typical sample shows:

	Per Cent.
Silica.....	48.0
lime.....	25.8
Iron oxide.....	4.3
Alumina.....	12.9
Carbonic acid.....	5.0
Undetermined.....	4.0
	100.0

Much analytical work is needed to determine the composition of the original unaltered limestone. While I am not prepared to discuss thoroughly the phenomena involved, my impression is that the original limestones were calcitic, and that ascending heated waters, following the intrusion of the monzonite, decomposed its ferro-magnesian members, deposited the iron as a replacement of the limestone near the contact, and dolomitized the limestones between the contact and the dike *BC*. But the reason of the different action east of the dike *BC* I leave to those better qualified to determine. In the east area are a number of vertical faults, some of great length and some short, nearly all of them showing a slight mineralization, chiefly of copper minerals in calcite veins, the gold and silver contents being negligible. Unfortunately the work done on these veins has not been productive; otherwise a fuller study would, perhaps, have been made of the conditions I have indicated above. The only working mine in the immediate region is situated 9 miles north-east of Fooshay Pass on the same contact, modified by some pegmatite dikes, at what was formerly known as the Bonanza King mine, where several million dollars worth of silver ores were extracted from irregular bodies in the limestones.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

The Furbero Oil Field, Mexico

BY E. DEGOLYER*, A. B. NORMAN, OKLA.

(San Francisco Meeting, September, 1915)

I. INTRODUCTION

THE Furbero oil field lies in the Gulf coastal plain of Mexico between the important ports of Tampico and Vera Cruz and about one-third of the distance between the two points from the latter (Lat. 20° 22' N., 97° 30' W. Gren.). It is in the State of Vera Cruz, about 14 miles west by south of the village of Papantla in the canton of the same name, and is the most southern of the developed oil fields of the Tampico-Tuxpam region being a distance of 50 miles south by east from the Potrero del Llano field and 42 miles in the same direction from the Alamo field. The field has been developed by the Oil Fields of Mexico Co. and is in the midst of a large block of lands owned or leased by them. Communication is through the port of Tuxpam with which it is connected by a narrow-gauge railway and 6-in. pipe line some 52 miles in length.

This field is of more than ordinary interest since it represents a type of oil deposit hitherto unknown. So far as I am aware, no other commercially valuable deposit of oil is known to occur under conditions of accumulation similar to those found here. The igneous nature of the so-called 'sand' encountered in drilling was not recognized until a number of wells had been drilled; and it was not until recently that the geological relations of the sill or laccolith, which gives rise to the accumulation of oil, were known and the so-called 'anticline' was known to be, in part at least, the result of igneous intrusion.

Geological surveys in this general region were made in 1900-1901 by W. H. Dalton and in 1907-1909 by P. C. A. Stewart, F. W. Moon, Leonard V. Dalton, and Dr. Max Muhlberg. More recent surveys have been made, however, under the direction of the author, by E. L. Ickes, B. C. Belt, B. Hartley, and Harve Loomis. Notes of the earlier examinations were not available with the exception of the report by Dr. Muhlberg who apparently first recognized the true nature of the Furbero intrusion. This paper is indebted to his work and that of the more recent surveys for much of the detail regarding stratigraphy and to Messrs. Percy Furber and Arthur C. Payne, President and General manager of the Oil Fields of Mexico Co. for permission to publish, and for much of the in-

*Chief Geologist, Cia. Mexicana El Aguila, S. A., Cuban Oil Co.

formation regarding the field. The detailed structure of the intrusion as herein set forth is the result of a study of the well logs and samples by the author.

II. HISTORY AND DEVELOPMENT

This general region, including Furbero, is one of the earliest known oil regions of Mexico. Indian tradition says that this part of the coast was settled during the seventh or eighth century; the Sabaneta topographic basin, in which Furbero lies, being then known as Choloa. That the existence of oil seepages was probably known to the early inhabitants,

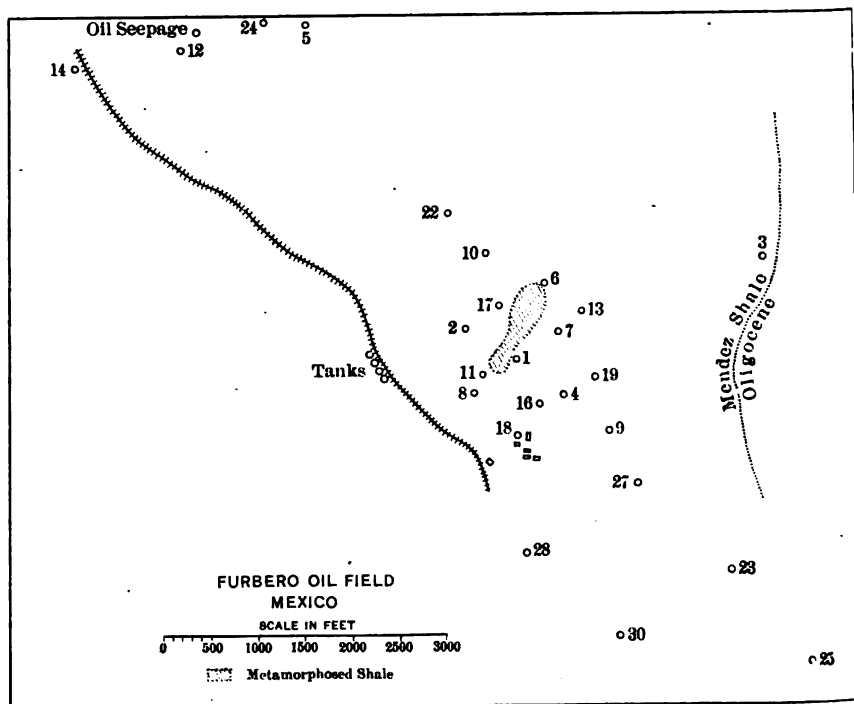


FIG. 1.—MAP OF FURBERO OIL FIELD, SHOWING DEVELOPMENT.

is indicated by the fact that the locality now known as Furbero was formerly called "Cougas" (later corrupted to "Cubas"), a Totonaco word, meaning a species of black wax or oil. Moreover, Spanish writers immediately following the conquest note "chapapote"¹ as among the

¹ "Chapapote" is the term by which asphalt, petroleum residue, and even oil itself are known in Mexico at the present day. It is derived from the Aztec words "tzauc," meaning paste or cement and "popochili," meaning perfume and probably refers to two of the early uses of the material. Robelo, Cecilio G.: *Diccionario de Seudazlequismas*, p. 536 (Cuernavaca, 1906).

articles for sale in the great market of Mexico City. Old workings consisting of a shaft walled up with rock were found near one of the principal seepages by the earliest known white explorer.

In 1868, a certain Dr. Autrey, who had been carrying on the practice of his profession at Papantla for several years, discovered the larger seepages, and the knowledge of their existence and location being made public, a company (the Cia. Explotadora de Petroleo del Golfo Mexicano) was formed at Mexico City in 1869, for the purpose of exploiting the deposits. Machinery was brought into the region and a 3-in. bore-hole was sunk to a depth of 125 ft. near one of the largest seepages. No production resulting, a tunnel was driven 50 ft. into the hill near the old Indian workings, and a moderate supply of oil (reported as about 4 or 5 bbl. per day) was secured. The project was soon abandoned, however, and about 1870 the properties, including abandoned machinery, were denounced by Autrey. One of the old stills was set up in Papantla and the crude oil was packed mule-back to that point where some 4,000 gal. of kerosene were refined and sold locally.²

In 1883, the present mining law, vesting the ownership of petroleum rights in the ownership of the surface of the land, was enacted, and the property thus passed out of Autrey's possession. In 1892, El Cougas and 2,500 hectares of land were leased to an American oilman, this being the first oil lease of record in Mexico.³

From 1895 to 1900, numerous leases were made in this general region, and in 1901 the Mexican Petroleum and Liquid Fuel Co., Ltd. was formed and took them over. This company was founded with English capital, among the interested persons being Cecil Rhodes of South African fame. During the years 1901 and 1902, drilling was actively carried on, 24 wells being drilled, two of which attained a depth greater than 1,500 ft. Operations being unsuccessful from a commercial standpoint, the region was abandoned. In 1903, the Oil Fields of Mexico Co., the present owner of the Furbero field, was organized. Lands and leases over a great area were secured. Machinery was brought into the region via the Tecolutla River and wagon roads; and in 1904 drilling was commenced in earnest. In 1907, the first well "came in"; and in April, 1910, the pipe line and railroad to the port of Tuxpam having been completed, commercial deliveries of oil began. Since that time drilling and oil production have been almost continuous. Thirty-two wells have been drilled by this company in the general region, 24 of them in the Furbero field proper.

² This early history is taken from a copy of a letter dated Mar. 2, 1892, from Adolpho Autrey, M. D., to John F. Dowling, City of Mexico.

³ Cuestiones Petroleras de Actualidad, Mexico, 1912; Lic. Edward Schuster, p. 1.

III. TOPOGRAPHY

The region in which the Furbero field occurs is topographically a great basin shut in on the north, west, and south by the great lava-capped mesas which lie at the foot of the central Mexican plateau (the remnants of a flow which once covered the entire region), and on the north and east by Tertiary hills. The major axis of this basin runs northwest and southeast and the drainage is controlled by two master streams: (1) The Cazones River, which emptying into the Gulf of Mexico about 40 miles northwest of Furbero, drains the northern part of the basin, including the Furbero field proper, through its tributary the Arroyo Coachepa and secondary streams; and (2) the Tecoluta River, which empties into the Gulf at a point 30 miles east of Furbero, draining the southern part of the basin. The water-sheds of these streams are separated by a very low divide. The Furbero field proper, as developed hitherto, has an area of little more than 100 acres. It lies in the northeastern part of the basin on the slope from the Tertiary hills to Arroyo Coachepa on the west. The most striking features of the immediate topography are the sharp twin hills in the midst of the field, which resemble very much the small volcanic plugs or necks so commonly features of the topography of the oil regions north of the Tuxpam River. The general elevation is from 600 to 650 ft. above sea level.

IV. GEOLOGY

This region is covered, for the most part, with dense tropical jungle and a mantle of soil resulting from centuries of vegetable decay; and as a consequence there are but few rock exposures of geological value. These are commonly found in the banks of creeks or arroyos. No exposures large enough to admit the measurement of detailed sections of the various formations are found. We are therefore forced to depend mainly upon the soil and topographic expression, verified by occasional exposures, in mapping the surface geology, and upon well records and general regional observations in studying the details of the various formations.

In considering the Furbero field proper, we are directly concerned with the Tertiary marine sediments and the igneous rocks only; but a description of the Cretaceous limestones is here given for the purpose of defining the relation of the Furbero field to other fields of the region.

1. *Cretaceous*

Tamasopo Limestone.—The oldest known rocks of this general region consist of a series of limestones, known from their outcrop in the Tamasopo Canyon, State of San Luis Potosi, as the Tamasopo limestone. This

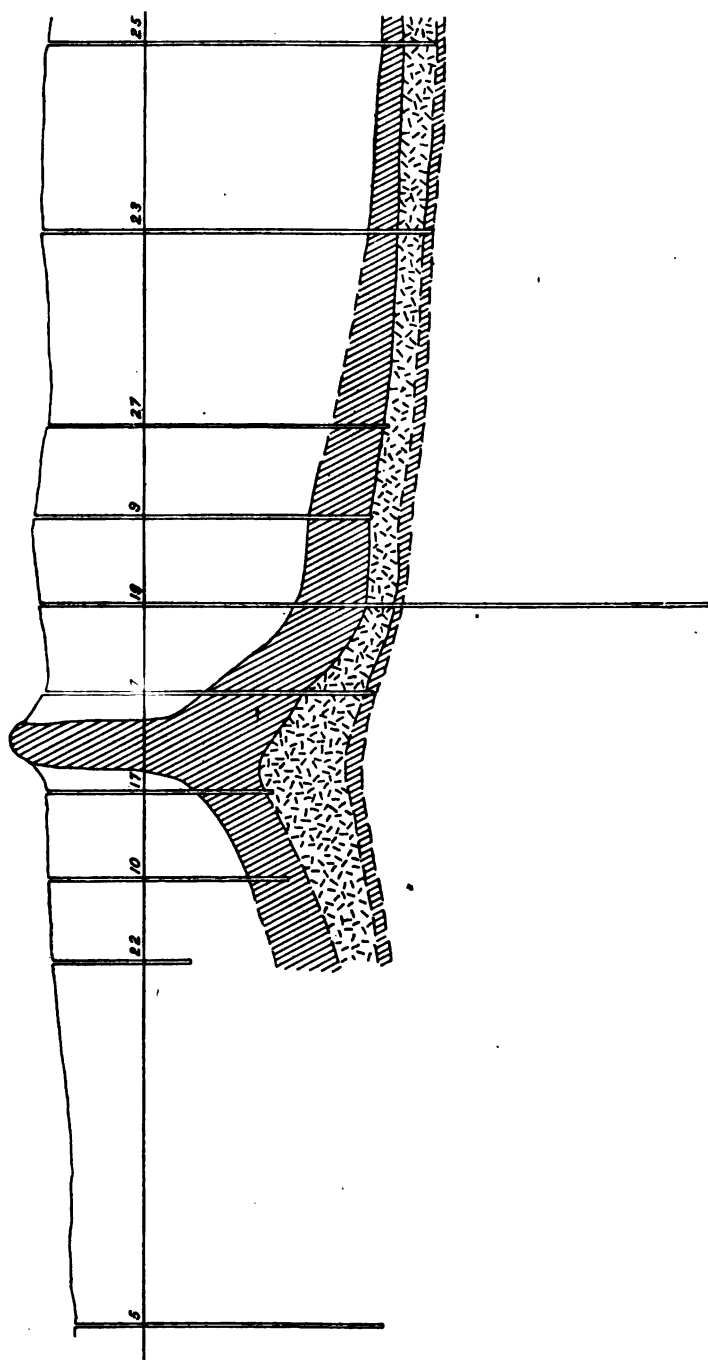


FIG. 2.—GEOLOGICAL SECTION OF THE FURBERO OIL FIELD.

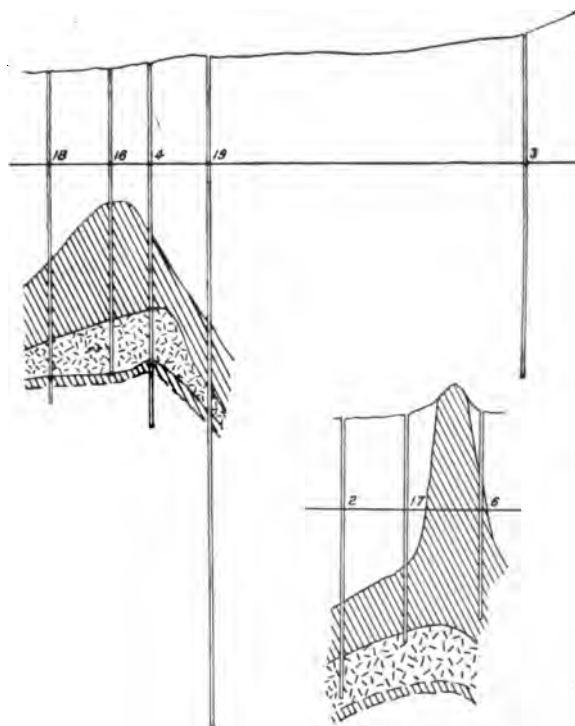


FIG. 3.

FIG. 4.

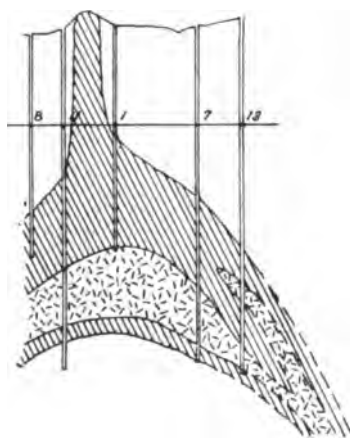


FIG. 5.

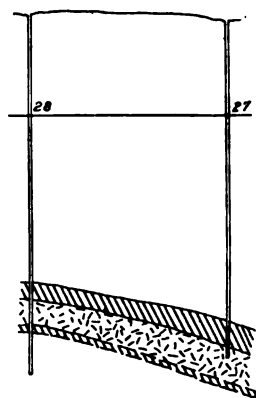


FIG. 6.

FIGS. 3 TO 6.—GEOLOGICAL SECTIONS OF THE FURBERO OIL FIELD

formation consists of hard, gray, pure, compact, porcelain-like limestones bedded in layers less than a foot thick, and is characterized, in its upper part, by the occurrence of an abundance of black to dark gray and green chert nodules, interbedded with the limestones. Economically, the formation is of greatest importance because of its oil-bearing horizons. The uppermost members of the limestone, which are massively bedded in the northern Vera Cruz and Valles regions, are somewhat porous and contain great solution caverns. These cavernous limestones are the reservoir rocks of highly productive fields of Potrero del Llano, Juan Cassiana, Dos Bocas, Los Naranjos, and Alamo, which are characterized by immense and prolific gushers (daily capacity 50,000 to 100,000 bbl.). The formation is generally regarded as the parent oil-bearing formation of northern Mexico, from which the oil at present found in other formations has come by migration.

The main mass of the outcrop of the formation in this region is in the Sierra Madres, the front range of which passes points respectively 28 miles west and 16 miles south of the Furbero field. The formation has not yet been encountered by wells drilled in this region, but it is very probable that the oil of this field originated in the Tamasopo and was forced upward by hydraulic and gas pressures, along a channel formed by the broken and metamorphosed zone of contact of igneous and sedimentary rocks, or through faults and fissures, to the reservoir from which it is now obtained.

The thickness of the formation in this region is unknown, but probably is from 6,000 to 8,000 ft. The age of the formation is lower Cretaceous, evidently the equivalent of the Comanche of Texas.

2. *Tertiary Eocene*

San Felipe Beds.—Overlying the Tamasopo limestone and resting unconformably(?) upon it, is a series of alternating, impure, thin-bedded limestones and gray, red, and green shales and marls known as the San Felipe beds. The entire formation is somewhat sandy and contains, locally, beds of tuff of variegated colors which contain decomposed mica and are finely porous. It is also reported to contain conglomerate though well records do not show such a rock. The economic importance of the formation in this region has not yet been determined. It has been encountered by two of the wells drilled and oil and gas have been produced from it, but not in great quantities. In the fields of the Panuco River valley near Tampico, it contains the oil-producing horizons. With the exception of one or two doubtful inliers, the outcrop of the formation in this region is confined to a narrow strip adjoining the outcrop of the Tamasopo limestone in the mountain front. The thickness of the formation varies from 600 to 1,000 ft.; it is probably more nearly the latter in this region. The exact age of the formation is unknown but it is apparently

Tertiary if one may judge from the few fossils which have been secured from drill cuttings. If such is true, it is of lower Eocene age. The formation grades imperceptibly into the overlying shales series, the limestones gradually becoming more argillaceous and impure and grading finally into hard shale and, in turn, into soft shale. It is doubtful if the upper 100 to 200 ft. of the formation can be differentiated from the overlying shales in well logs.

Mendez Shale.—Grading from the underlying San Felipe beds, is a thick series of gray to green shales, marls, and clays, containing rarely thin shaly sandstones and limestones and red shales, called the Mendez shale, because of its outcrop near Mendez station, just west of Tampico. Economically, it is generally of no importance; but in the Furbero field, part of its ordinarily impervious shales have been baked and metamorphosed into a hard, brown to black, porous shale and it forms, together with the crystalline igneous rock the oil-reservoir rock of the field. Apparently much of the metamorphism has been due to the action of ascending thermal waters associated with the igneous rock after its intrusion; for the pipe of metamorphosed shales which rises to the surface and outcrops in the twin hills in the central part of the field cannot have been the direct result of the heat of the igneous rock. The igneous rock of the intrusion has also undergone alteration since crystallization.

This formation outcrops for the most part over the entire floor of the Sabaneta basin. It is the surface rock of the Furbero field proper, extending from the Oligocene hills on the east to the lava flows at the foot of the hills of the Sierra Madre on the west. The thickness of this formation at Furbero is approximately 4,000 ft. No fossils have been found in this region; but a collection made at Alazan some 50 miles to the north has been identified by Dr. Dall as lower Eocene.

3. Tertiary: Oligocene

Overlying the Mendez shales is a thick series of sandstones, shales, impure fossiliferous limestones and occasional conglomerates of Oligocene age. The various strata making up the formation are lenticular and grade laterally into each other. Near the front of the Sierra Madre occur beds of shale so thick that their outcrops are hardly distinguishable from those of the Mendez shales. The formation is not known to be of economic importance in this region. The local outcrop occurs over an area east of a N-S line just east of well No. 3.

4. Igneous Rocks

The igneous rocks of the region consist of basalts, dolerites, basalt-gabbros, and various products of volcanic activity, such as volcanic sands, ash, etc. The surface outcrops consist almost altogether of lava-capped mesas, the remnants left by the erosion of a great lava flow which once

covered most of the region. Several big dikes are found near Reparo and many porphyritic basalt dikes cut the Oligocene rocks near Buena Vista. A patch of volcanic ash covers the surface near Palma Sola. The lava flows evidently originated at some point in the Sierra Madre, probably in the vicinity of Necaxa. The flow varies in thickness from a few feet to 500 ft., and it has a uniform inclination of several degrees to the northeast, an inclination due evidently to the topography of the pre-existing land surface. The nearest remnants of this flow to the Furbero field are the lava-capped mesas of El Brinco, 6 miles to the northwest, Mecatepec, 9 miles north, and a mesa near Tecuantepec River, 12 miles east of south. There are no evidences of deformation since these flows, which occurred after the deposition and folding of the entire series of marine sediments.

Of greater interest and economic importance is the occurrence of a laccolith or extremely thick sill, the formation of which, together with associated phenomena, has given rise to the accumulation of oil known as the Furbero pool. This intrusion does not reach the surface. The only indication of its existence is the outcrop of metamorphosed shale already mentioned. It has been well explored, some 16 wells having been drilled into it, five other wells into the overlying metamorphosed shales, and six wells, encountering no evidence of intrusion, in the immediate vicinity. The intrusion evidently commenced in the form of a sill following closely the bedding planes, the sedimentary rocks having already been folded into an anticline. The intrusion thickened near the crest of the anticline, lifting the overlying beds and further accentuating the already existing fold. Small amounts of oil are produced from the igneous rock in several of the wells. As has been observed, the gabbro seems to have been greatly altered and is quite porous in certain localities. The greatest thickness of igneous rock shown by the drill is in well No. 7 where 440 ft. were drilled through. A careful construction of the probable form of the intrusion, however, suggests a maximum thickness of from 600 to 650 ft. near wells 17 and 7. Minimum thicknesses of 193 ft. and 170 ft. have been encountered in wells 19 and 28 respectively, though the actual thickness of the intrusion is probably little more than 100 ft. at well 19; the greater apparent thickness being due to the fact that the intrusion is dipping steeply where encountered by the bore, which therefore does not show a true section. The highest point reached by the intrusion, as shown by well records, is in well No. 17 where the top of the intrusion was reached at a depth of 1,490 ft. or 891 ft. below sea level. The lowest point yet reached is in well 25 at a depth of 2,665 ft. or 1,984 ft. below sea level. The general form is indicated by the accompanying sections. Two sills of igneous rock separated by some 300 ft. of metamorphosed shale were passed through by well No. 13. The area known to be underlain by the igneous rock as determined by wells which have

been drilled up to the present time (encountering either the igneous rock itself or the metamorphosed shale) is 26 acres; but the actual area underlain is known to be greater than this, probably several times as large.

5. *Structure*

The general structure of this region, as of the Gulf coastal plain for some distance northward, is that of an eastward-dipping monocline; the oldest rocks—the Cretaceous limestones—outcropping in the steeply folded and faulted ranges of the Sierra Madre and passing under the Tertiary rocks of the coastal plain, and the Tertiaries outcropping generally in the order of their succession, the youngest beds fringing the coast. The beds of the monocline are gently folded, the complexity and steepness of the folding and faulting being greatest in the Sierra Madre, and decreasing with distance from it. Folding took place in Oligocene or post-Oligocene time and afterward the entire region was subject to igneous activity. As has been noted, it was almost covered by an immense lava flow, the remnants of which cover hundreds of square miles at the present time. The sedimentary rocks were also intruded to some degree; but the surface evidences of intrusion are not very extensive.

The general structural strike of this entire region is northwest-southeast, the axes of the minor folds of the Tertiary rocks of the coastal plain being parallel to those of the steeply folded Cretaceous rocks of the Sierra Madre. One of these minor folds, the Furbero anticline, having been complicated by the intrusion of igneous rocks and the metamorphism of previously impervious beds, affords the structure in which the oil of the Furbero pool has accumulated. It may be held by some that the folding of the sedimentary rocks has been due to the intrusion. That they may have occurred contemporaneously is possible; but that the intrusion represents an incident of the more general igneous activity which took place in the Tampico-Tuxpam region after the folding of the sedimentary rocks, seems more probable. That the intrusion followed generally pre-existing structure seems probable. The major axis of the Furbero anticline and the major axis of the intrusion are the same and parallel to the axis of the other folds in the general region. The floor of the intrusion is of a shape generally anticlinal and is evidently influenced in form by conditions existing at the time of the intrusion. Had the strata been level when the igneous rock was intruded, one would have expected to find the laccolith with a flat floor, the arching of the beds being confined to those above the plane of intrusion. The dip of the upper surface of the laccolith, though in the same direction as that of the floor, is generally steeper; but the exact effect of the intrusion on the position of the outcropping strata cannot be determined because of scarcity of surface ex-

posures. It seems probable that the intrusion ascended from below in the form of a dike, and encountering a bedding plane of little resistance which was dipping steeply (west flank, sections *B* and *C*) was diverted into that plane, and took the form of a sill. The splitting of the intrusion (well 13) probably indicates the point of this change of form.

V. DETAILS OF DEPOSIT

As will be noted from the accompanying map, the laccolith has been intensively explored, several of the wells being only 80 to 100 ft. apart and most of them being within 200 ft. of other wells, so that several have been found to interfere with each other in actual exploitation. Wells more recently drilled in the exploration of the southern part of the laccolith have been located from 800 to 1,000 ft. apart.

This field is structurally a somewhat complicated anticline, and the occurrence of oil may be classed as conforming to the structural theory; yet the determining factor in the accumulation of oil has been the providing of pore space for a reservoir by the metamorphism, accompanying the igneous intrusion, of the under- and overlying shale, which were previously impervious. Of consequence also has been the subsequent alteration of both shales and igneous rock; but, as this was probably an indirect result of the igneous activity, it need not call for separate consideration. Normally, the unaltered sediments were probably quite similar to the unaltered shales into which they grade, both the altered and unaltered shales being of the Mendez formation, a series of blue and gray, medium-soft, fine-grained shales, more or less calcareous in places, and (when not metamorphosed) of fairly constant lithological character throughout. The metamorphosed shales seem to be of two general types: A fairly hard, black shale, differing but little from the unaltered gray shales into which it grades; and a hard, brown, porous, silicified shale, which usually occurs immediately above the igneous rock and grades into the black shales. The zone of metamorphosed shales envelopes the igneous rock which it both overlies and underlies, and is distinctly thicker over the intrusion than under it. Above the apex of the intrusion the metamorphosed zone continues in the form of a pipe or chimney to the surface, where its outcrop forms two sharp hills, resembling in contour the volcanic plugs of the region north of the Tuxpam River. This extension of the metamorphosed zone is probably due to ascending gases or thermal waters, and not directly to the heat of the intruded masses. As is seen on the sections, the upper metamorphosed zone including both the black and the brown shales, varies from 100 to 500 ft., and the underlying zone from 50 to 150 ft., in thickness.

The igneous rock is a dolerite or gabbro which has undergone, in places a somewhat extensive alteration. Drill cuttings of the rock consist for

the most part of gray sand much resembling a true quartz sand. This sand, together with the position and structure of the igneous rock, which somewhat resembles that of a thick lens of folded sedimentary rock, was quite conducive to concealment of the true igneous nature of the intrusion; and the geology was first interpreted as that of a simple fold in sedimentary rocks. Pieces of igneous rock several centimeters in diameter have been recovered from the cuttings. A sample from the intrusion penetrated by well No. 13 at 2,400 ft., has a porosity of 6 per cent., as determined by experiment. A sample from 1,835 ft. in well No. 16 is apparently a coarsely crystalline dolerite and is quite porous. A characteristic log of a well in this field as partly determined from well samples, is as follows:

Furbero Well No. 11	Thickness, Feet	Depth, Feet
Yellow clay (surface).....	29	29
Blue and gray shales with oily seams, between 400 and 900 ft. (unaltered).....	1,221	1,250
Black and brown shales of variable hardness (metamorphosed) with good shows of oil at 1,650 and 1,655 ft.....	415	1,665
Gray sand (igneous crystalline; good show of oil at 1,650 ft. contact).....	363	2,028
Black shales (metamorphosed).....	122	2,150
Gray shales (unaltered).....	200	2,350
Small production of oil at the upper contact zone of igneous and sedimentary rocks.		

VI. OCCURRENCE OF THE OIL

The oil in this field occurs in commercial quantities both in the igneous and metamorphosed rocks. Well No. 27, the largest well in the field, though it was already producing while in the upper metamorphosed shales, "came in" with an initial production of almost 1,000 bbl. per day upon being drilled 30 ft. into the igneous rocks. It has produced more than 200,000 bbl. of oil, most of which has apparently come from the igneous-rock reservoir. Well No. 28 also showed an initial production of 150 bbl. per day from a depth of 150 ft. into the igneous rock. Well No. 9 has produced approximately 150,000 bbl. from the upper metamorphosed shales; and most of the production of the remaining wells has come from the metamorphosed shales, the zone of contact with the igneous rock being particularly productive. Salt water has been encountered immediately under the oil in several of the wells.

As has been observed, the greatly varying porosity of the rock, rather than the structure, is believed to have been the determining factor in the accumulation of the oil of this deposit. The most productive wells of

the field have been Nos. 9 and 27, and they lie to the east of the center of the area of greatest production. The crest of the laccolith or dome lies altogether outside the most productive area, though some oil has been encountered in every well drilled to a depth at which it might reasonably be expected. In my opinion, the intrusion would have been as effective in any form which it might have assumed, provided that it did not outcrop.

From a knowledge of this deposit and of the Mexican fields in general, I believe that the oil of this deposit originated in the Cretaceous limestone, and has been forced upward to its present position along the metamorphosed zone of contact of the sedimentary and igneous rocks.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Manufacture and Tests of Silica Brick for the Byproduct Coke Oven

BY KENNETH SEAVER,* S. B., PITTSBURGH, PA.

(San Francisco Meeting, September, 1915)

INTRODUCTION

It is rather gratifying that in the field of the manufacture of refractories, in which, as a whole, foreign practice has surpassed us, the United States stands pre-eminent in the making of silica brick. In some instances European manufacturers have refused point-blank to credit the possibility of making to such specifications as are common with us, until the actual completed shapes were shipped for inspection. But both the manufacturer and consumer in foreign lands are now frankly learning from us, and benefiting by our advanced practice in this field, as is evidenced by the orders placed here and the efforts put forth abroad to come up to the standard of American-made material.

In no metallurgical operation has the employment of silica brick spread with such leaps and bounds as in the building of plants for the carbonization of coal, particularly in the byproduct coke oven. Today, in practically all types of oven, the structure above the floor level (except the facings) is of silica, while in the recuperative and in some regenerative types, its use is extended to that portion of the oven below the floor as well.

By reason of the fact that there are a number of classes of refractories of higher silica content than the usual so-called fire-clay brick, there is sometimes a misunderstanding as to the class of material referred to as "silica" brick. We here refer only to a brick having a silica content of 94 per cent. or more and made usually from quartzite rock with a small percentage of lime as a binder. The term "silica" brick is sometimes rather loosely applied to brick from highly siliceous clays, or to a quartzite refractory of mingled fire-clay and quartz rock; but such products are not considered in this discussion.

MATERIALS

The rock from which practically all high-grade silica brick are made is a true quartzite or metamorphosed sandstone, variously known as

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ganister, quartz, quartzite, silica rock, etc.—the largest fields, in the order of their relative importance being those of Pennsylvania, Wisconsin, Alabama and Colorado.

Pennsylvania Quartzite

Pennsylvania manufactures probably from 75 to 85 per cent. of all the silica brick made in the United States; nearly all are made from the quartzite found in the Medina and Oneida formation. While this formation extends across the State of Pennsylvania reaching into New York State on the north, and Maryland, Virginia and still farther toward the South, the quarries are practically all in Huntingdon and Blair Counties, Pennsylvania. It is in this restricted area that the quartzite is at its best. Toward the north and toward the south it becomes softer, is no longer a true quartzite, or is too much impregnated with iron. The rock is also comparatively easily accessible in the counties mentioned, enormous quantities lying loose upon the surface. The particular occurrence of this loose rock has here been brought about by the Juniata River and its tributaries, as they have cut across the formation, wearing away the softer formations beneath, and leaving the Medina overhanging and exposed. In the course of time it has been weathered and broken up by the elements into the bodies of loose rock or "floes," as they are called, which may cover an area of a thousand acres or more. The depth usually varies from three or four to as much as 20 ft. Smaller floes formed in the same way may be found along the flanks of the mountains; but they nowhere compare in size with those formed where the measures have been cut transversely.

This rock is sometimes quarried out of the solid measures, but by far the largest amount used is that taken from the bodies of loose rock, as the latter is not only cheaper to work but is of more uniform quality. The white Medina formation is not composed entirely of quartzite, but is made up principally of sandstone with intermingled strata of quartzite. Thus where the formation has been broken up, the softer sandstones unsuitable for the manufacture of silica brick have tended to weather away and disintegrate, thus leaving the larger portion of the floes consisting of quartzite. When sandstone is mixed with the quartzite in the floes it is readily detected by its appearance, as it tends to become much more rounded and worn in weathering than does the quartzite, which maintains its sharp angular edges.

The individual pieces in these floes range from those of fist-size to boulders weighing many tons. Probably 50 per cent. of the rock occurs in pieces weighing more than 500 lb. The largest floes are found at the gorges of the Juniata River near the towns of Mount Union, Mapleton, Point View, Lewistown, etc.

It is interesting to note that the Oriskany sandstone, extensively used as glass sand, is found in close proximity to the Medina and that although exceedingly pure in its composition, its softness renders it unsuitable for high-grade silica brick. It consists of a mass of minute quartz grains loosely held together, and often so friable that it can be easily worn away by the hands. On the other hand, the Medina quartzite consists of quartz crystals cemented in a mixture of the same composition and of equal hardness with the crystals. The difference in the two types of rock when ground to sufficient fineness for the manufacture of brick is obvious. The harder stone gives splintery, angular fragments; the other, rounded grains. Experience shows that a hard rock like the Medina is required for a physically strong silica brick.

The white Medina which occurs in Southern Pennsylvania near the town of Hyndman is there prominent and of a pure variety, but by no means as hard as the quartzite of Huntingdon County.

A representative analysis of the Pennsylvania quartzite is as follows:

	Per Cent.
Silica (SiO_2).....	97.80
Alumina (Al_2O_3).....	0.90
Ferric oxide (Fe_2O_3).....	0.85
Lime (CaO).....	0.10
Magnesia (MgO).....	0.15
Alkalies.....	0.40
	100.00

Wisconsin Quartzite

This rock, much used in the manufacture of silica brick in the vicinity of Chicago, is known as the Baraboo quartzite from the town of that name where these measures appear, and near which the quarries are located. The rock occurs in high cliffs throughout this region and is especially prominent around Devil's Lake. Here glacial action has excavated a broad deep valley in the quartzite formation bounded on each side by cliffs several hundred feet in height. Their bases are covered with the loose rock split from the parent ledges. While the fragments are much larger as a whole than the weathered Medina of Pennsylvania, the quality by no means compares with the latter. Practically all the stone used in brick making is here quarried from the solid measures.

The marked peculiarity of this rock, as compared with the Pennsylvania quartzite, is its reddish color, which ranges from a light pink through a dark purplish red almost to black. It is often as red as the red Medina of Pennsylvania, a sandstone wholly unsuited for brick making. While the coloring of the Baraboo quartzite is due to iron, it is yet an exceedingly pure rock. The quartzite often becomes lighter in

color as the quarry works in from the face of the cliff, but it is the exception to find white rock.

This Baraboo quartzite, estimated as some 4,000 to 5,000 ft. in thickness, is a metamorphosed sandstone, the sedimentary origin of which is indicated by its parallel bedding planes, by ripple marks, etc. Layers of quartzite are often separated by beds of quartz schist varying from a few inches to several feet in thickness. This schistose structure is due to the admixture during deposition of clayey material, which in the subsequent folding and bending of the rock has developed a slablike or schistose structure.

The United States Geological Survey explains the formation of the quartzite as due to the enlargement of the original quartz grains and to the deposition of interstitial quartz. It is the latter that carries the iron oxides, which give the characteristic color to the rock. Regardless of its color, the formation throughout shows the physical characteristics of hardness and strength, as well as the splintered and angular fracture in crushing, identical with the Medina quartzite of Pennsylvania. A typical analysis follows:

	Per Cent.
Silica (SiO_2).....	97.15
Alumina (Al_2O_3).....	1.00
Ferric oxide (Fe_2O_3).....	1.05
Lime (CaO).....	0.10
Magnesia (MgO).....	0.25
Alkalies.....	0.10
	99.65

Alabama Quartzite

The material used in silica brick manufactured in Birmingham, Ala. is found in the eastern part of that State. It is bluish white, and as hard and crystalline as the Wisconsin quartzite; average analyses show it to be equally pure. It occurs in lenses of varying size, in a sandstone formation. It is probably a quartzite formed by a leaching out of impurities from the parent rock followed by a crystal growth and additional deposit of silica from circulating waters. A typical analysis follows:

	Per Cent.
Silica (SiO_2).....	97.70
Alumina (Al_2O_3).....	0.96
Ferric oxide (Fe_2O_3).....	0.80
Lime (CaO).....	0.05
Magnesia (MgO).....	0.30
Alkalies.....	0.31
	100.12

In Colorado and Montana both quartzite and sandstone are used to some extent in the manufacture of silica brick.

METHOD OF MANUFACTURE

Quarrying

The methods of quarrying as necessitated by the mode of occurrence of the quartzite are so obvious as to need no description. A point that will bear emphasis however, is the necessity of adequate supervision in the selection of the rock, whether it be from floes or from the solid measures. Whatever the location or occurrence of the rock, such care is essential; the easiest way to keep bad rock out of the brick is never to let it reach the plant.

Grinding

After the rock broken to one man size is received at the rear of the brick plant, it is put through some one of the familiar types of crushers, which reduces it to practically the size of a 1- to 2-in. ring. It is then elevated to storage bins from which it may be fed direct in definitely weighed quantities to the wet pans for grinding.

Into the wet pans, as the grinding proceeds, 2 per cent. of lime, by weight, is introduced as milk of lime. When the brick is burned, the lime in combination with the other elements in the mass serves to give the necessary bond or physical strength. The quality of the lime has no small effect upon the resulting characteristics of the brick. The effect of an increase in lime from 2 to 3 per cent. is not marked; but such as it is, renders the brick less able to withstand abrasion, as is indicated by rattler tests. The decrease in strength on further additions of lime up to 7 per cent. is progressive, and becomes more and more evident with each addition. There is, on the other hand, a decrease in strength at 1.5 per cent. of lime, and a marked weakness and friability as the amount of lime is still further diminished. As regards ultimate refractoriness, no marked effect is noticeable upon increasing the lime addition up to 3 per cent. From that point on, the effect is clear in a lowering of the fusion point.

The fineness to which the charge in the wet pan is to be ground depends upon the nature of the brick to be made from it and is, of course, controlled by the time during which the charge is allowed to remain in the pan. In general, one charge, containing material for 200 brick, will be ground at the rate of approximately four pans per hour for standard 9-in. brick, while to obtain the fineness requisite for the intricate and difficult shape-work necessary in byproduct coke-oven construction, but two pans per hour may be possible. The rock is exceedingly hard, and a comparatively slight increase in the fineness of grind is accompanied by a seemingly disproportionate lowering of grinding capacity, and increase in wear on the pans.

The actual differences in grind, however, are much smaller than is often imagined. Figs. 1 and 2 are photomicrographs with a magnification of 2 diameters, the ground material showing on a background of cross-section paper with 0.1-in. spacing. The specimens are from pieces of green or unburned silica brick of equal weight, crushed down after drying. Fig. 2 indicates typical fine-ground shape-work, while Fig. 1 is that of average standard 9-in. brick.

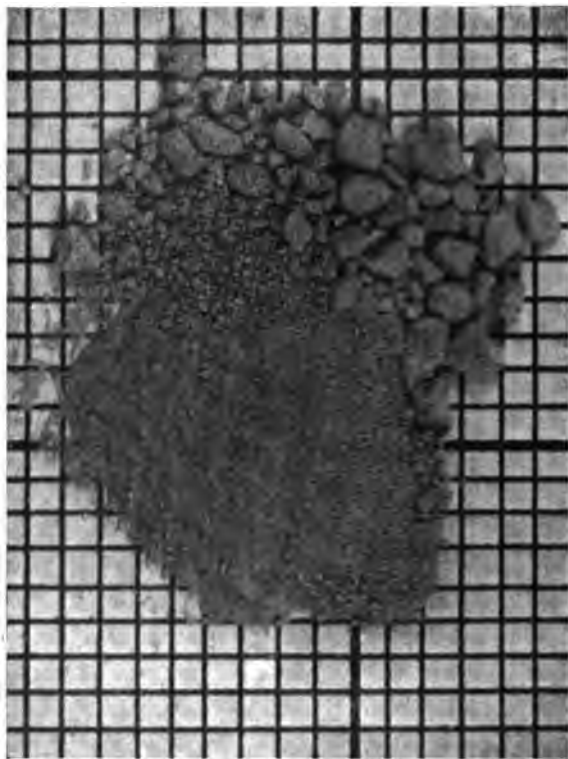


FIG. 1.—MATERIAL FROM UNBURNED AVERAGE STANDARD 9-IN. BRICK.
MAGNIFICATION 2 DIAMETERS.

It will be plainly apparent from the photomicrographs that the difference lies not so much in the actual size of either the coarse or the fine grains as in the relative proportion of fines. In the finer-ground brick the larger proportion of these is clearly evident. There can be no question that the general effect of increasing the fineness of grind is to improve the abrasive resistance of the brick, if it is properly burned. In general the requirements as to grind contained in typical coke-oven specifications have worked no hardship upon the manufacturer; for the present-day standard of workmanship on byproduct oven shapes is such

as to "carry its own grind with it," if the expression may be used. To secure the accurate filling out of the shapes, and the clean-cut accuracy everywhere essential, the grind is actually much finer than is requisite to satisfy the usual specifications as to that particular feature.

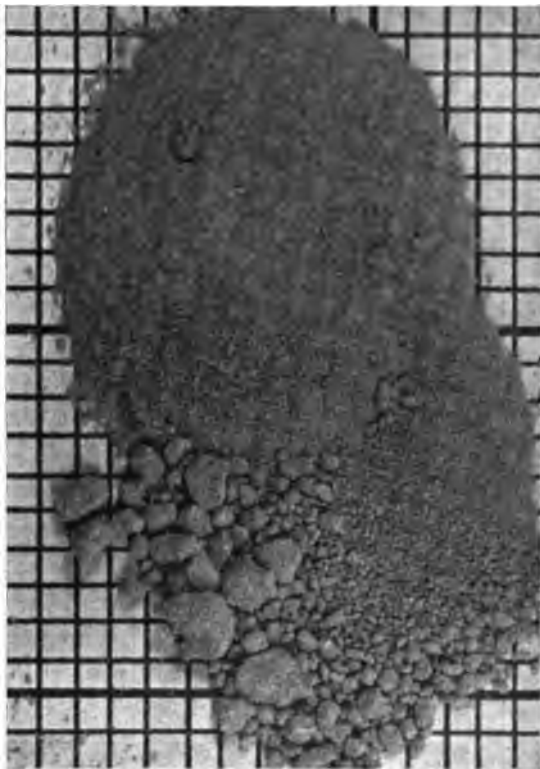


FIG. 2.—SAMPLE SHOWING MATERIAL USED IN TYPICAL FINE GROUND SHAPE WORK
MAGNIFICATION 2 DIAMETERS.

Molding

The ground silica material as ready to mold is of about the consistency of damp sand, but probably because of the extreme angularity of its grains has less flow as it is manipulated. As a consequence, the molds must be filled by pounding with heavy plank beaters covered with metal, and all corners and shapes of irregular outline must be rammed in by hand, much as sand molds are made, but with much heavier blows. Unequal ramming, which leaves too great a variation in the density of the green brick, will give unequal expansion in the kiln. It is by no means uncommon to lose hundreds of a new and difficult shape through such fire-cracking. To get proper molding and ramming requires the strictest supervision.

The wear on the molds due both to the attrition of the grains and the pounding is exceptionally severe. In view of the extreme accuracy required, steel molds are used exclusively for all high-grade coke-oven work. These molds are often decidedly complicated, and always expensive, as "loose sides" must be used throughout. Ground silica has, as molded, almost no physical strength, and can be taken from the mold only by allowing the sides to come with it, after which the sides are stripped.

After molding, the shapes are placed on pallets, on which the brick are dried either on the steam-heated hot floor or on rack cars in some form of tunnel dryer (Fig. 3). The brick are then ready to be set in the kilns. Although setting is a most important feature in the manufacture, no attempt will be made to discuss it here.



FIG. 3.—TYPICAL SECTION OF HOT FLOOR, SHOWING COKE OVEN "GUN" BRICK-SHAPES IN PROCESS OF DRYING.

Burning

The vast majority of silica brick are burned with soft coal in the familiar type of round, down-draft kiln, varying in diameter from 30 ft. upward. Although practice may vary somewhat, and much depends on the size of the kiln, the following schedule of burning may be considered as fairly normal. Assume that the setting is complete and the doors are sealed. Fire is immediately started, and continued slowly and with slight acceleration for from 9 to 12 days, during the last 2 or 3 of which the final temperature should be practically attained. The kiln is then held at the required temperature, say at 26 cone (1,650° C., or

3,002° F.) directly in front of the firebox, for another day, after which the fireboxes may be sealed and allowed to remain thus for approximately 24 hours more. Cooling will require about as long as the burning proper; but this varies somewhat with atmospheric conditions.

It is obviously difficult to take temperatures in the interior portions of the kiln except by cones; but there are often most perplexing variations in this method. There are apparently certain conditions in the atmosphere of a burning silica kiln which alter the readings of standard cones to no small degree. Cones which give concordant results, week in and week out, in clay kilns are often decidedly erratic in registering the temperatures of silica kilns. It is by no means uncommon to witness cones of higher number go down, while next to them those lower in the series are either but slightly bent or apparently unaffected.

Effect of Burning on Composition and Expansion

In any discussion of the desirable qualities of silica brick there is frequently expressed the necessity that "all the expansion shall be burned out of them." This brings out the following questions: What is the nature of this expansion; what produces it; what is its extent, and how fully can it be controlled.

That the terms used in the discussion may be clear, let it be premised that all silica brick are of course made in molds smaller than the proposed size of the finished or burned brick, and that under the influence of heat a permanent expansion occurs, swelling the brick from the mold or "green" brick size to the required or "burned" size. It is desired that this permanent expansion occurring during burning be completed so as to preclude further permanent expansion in subsequent use.

Now, in every burned silica brick expansion will occur on heating—a true thermal expansion, identical in its general character with that incident to the heating of practically all bodies. Such thermal or temporary expansion will disappear upon cooling, and being the expression of the coefficient of expansion of the silica it is unavoidable. This thermal or temporary expansion; occurring upon heating a burned silica brick is not to be confused with the permanent expansion produced in the body of the brick during its manufacture. It is argued that, if a silica brick is not properly burned, then, upon reheating in actual coke-oven practice, there will be not only a normal thermal expansion but an additional expansion of indeterminate amount which will be permanent. It is essential in the construction of the coke ovens that the expansion, whatever its nature, be as small as possible, and that, whatever it is to be, it be known and adequately cared for.

It will be recalled that the silica brick under consideration is formed by adding 2 per cent. of lime to quartzite rock of the following composition, say:

	Per Cent.
Silica (SiO_2).....	97.80
Alumina (Al_2O_3).....	0.90
Ferric oxide (Fe_2O_3).....	0.85
Lime (CaO).....	0.10
Magnesia (MgO).....	0.15
Alkalies.....	0.40
	100.20

The representative silica brick produced may be assumed to have the following composition:

	Per Cent.
Silica (SiO_2).....	96.25
Alumina (Al_2O_3).....	0.88
Ferric oxide (Fe_2O_3).....	0.79
Lime (CaO).....	1.80
Magnesia (MgO).....	0.14
Alkalies.....	0.39
	100.25

What are the changes which occur as the siliceous mass is burned?

Certain of these are obvious, such for instance as the formation of a bond of one or more of the lime-silica compounds such as the meta-silicate CaO , SiO_2 or the ortho-silicate 2CaO , SiO_2 . With the exception of the lime present, the ferric oxide of iron and the alumina, though small in quantity, are the preponderating impurities and there is undoubtedly formed a certain amount of silicate of iron as well as compounds of the lime-alumina series. Whereas the lowest melting point incident to any eutectic of the lime-silica system is $1,436^\circ\text{C}$. ($2,617^\circ\text{F}$.) and of the alumina-silica series is $1,610^\circ\text{C}$. ($2,930^\circ\text{F}$.), while that of the lime-alumina system is $1,400^\circ\text{C}$. ($2,552^\circ\text{F}$.), some of the eutectics of the ternary system CaO , Al_2O_3 , SiO_2 melt at approximately $1,170^\circ\text{C}$. ($2,138^\circ\text{F}$.) and are thus among the first of the flux compounds resulting.

No less important, though less recognized, changes are occurring in the brick as it is heated. The evidence of the change, regardless of its reason, is easily indicated by the most casual measurement when brick are drawn from the kiln from time to time, as higher temperatures are reached. The final expansion will be very close to $\frac{3}{8}$ in. per foot.

The theory of this expansion is that a change occurs in the crystalline form of the silica, an inversion from the quartz crystal to the crystalline form of cristobalite. The intermediate crystalline form of tridymite does not apparently occur here. Since these various crystal forms have varying specific gravities the volumes accordingly vary.

The valuable work done by Day and Shepherd¹ and particularly by Fenner,² give the results in a large field of laboratory experiment, which

¹ The Lime-Silica Series of Minerals, *American Journal of Science*, 4th series, vol. xx, No. 130, pp. 265 to 302 (1906).

² The Stability Relations of the Silica Minerals. *American Journal of Science*, 4th series, vol. xxxvi, No. 314 (1913).

make possible a fair understanding of what occurs in the actual burning of the silica brick. Fenner's work was to establish among other things the relations between the silica minerals quartz, tridymite, and cristobalite, any of which may be converted into any other of these by appropriate heat treatment. For the purposes of this discussion it is unnecessary to touch upon the alpha and beta phases of these minerals. In substance, the results under the conditions imposed by the experiments were that powdered quartz, if heated, remains quartz until a temperature of 870°C . ($1,598^{\circ}\text{F}$.) is reached, at which inversion point there begins a slow change to tridymite. If heat is still applied, there is no further change from tridymite until $1,470^{\circ}\text{C}$. ($2,678^{\circ}\text{F}$.) is reached, when inversion to the form of cristobalite has its beginning. There is no further change until the cristobalite melts into a glass. Thus the stable phase of quartz (quartzite rock as applied to silica-brick manufacture) extends to 870°C . ($1,598^{\circ}\text{F}$.). Between 870°C . ($1,598^{\circ}\text{F}$.) and $1,470^{\circ}\text{C}$. ($2,678^{\circ}\text{F}$.) the stable phase is tridymite, while above $1,470^{\circ}\text{C}$. ($2,678^{\circ}\text{F}$.) cristobalite is stable.

As a whole it must be carefully noted that the changes from one mineral to another were exceedingly slow, particularly near the inversion points. So slow in fact were the alterations, that the melts were made with a large excess of sodium tungstate which acted as a flux or catalyzer. The powdered quartz or whatever might be the silica mineral under study was stirred into the molten bath of flux and the platinum container was then heated in a suitable furnace. This flux in no way entered into chemical combination with the silica but only aided and quickened its action, apparently furnishing a fluid medium through which the molecular rearrangement was facilitated. In spite of this, the slowness of the inversion, often requiring many hours, will bear emphasis in drawing conclusions regarding any parallelism between these experiments and the burning of brick.

A particularly noteworthy fact is that in the absence of a flux or catalyzer, the inversion from quartz to tridymite does not occur, but the inversion is direct to cristobalite, and there is a shift in the inversion point to as low as $1,250^{\circ}\text{C}$. ($2,282^{\circ}\text{F}$.) although the change at this temperature is slow.

This entire absence of tridymite in quartz heated well above 870°C . ($1,598^{\circ}\text{F}$.) the quartz-tridymite inversion point, although difficult of explanation, is apparently proven. As indicative of the time element requisite for such an inversion, in one experiment fine-ground quartz heated 108 hr. at $1,250^{\circ}\text{C}$. ($2,282^{\circ}\text{F}$.) without flux gave but a small percentage of inversion while with 90 hr. at $1,360^{\circ}\text{C}$. ($2,570^{\circ}\text{F}$.) approximately two-thirds became cristobalite.

In considering these changes in the crystalline form of silica, some of the characteristic properties of the three will not only make clearer the method of distinguishing them, but also throw some light on the reasons

for the effects these inversions exhibit.

Quartz has a specific gravity of 2.65, tridymite of 2.27, and cristobalite of 2.33. Each has its characteristic crystal form, the quartz being usually in the familiar hexagonal pyramids, prisms or combinations of these. Tridymite often occurs in thin tablets or plates hexagonal in shape. Cristobalite is exceedingly irregular in the development of its crystal form but often develops octahedral caps along the closely interlacing axes of elongation.

It is the difference between the indices of refraction of these minerals, however, that furnishes the greatest aid in distinguishing the one from the other.

Tests.—As a starting point from which to work in determining the effect of burning, a set of sample silica brick were taken from one kiln, all from the same zone and from a kiln of absolutely normal burn. In this instance they were first burned at a point about 8 ft. from the wall and 6 ft. from the bottom. This sample was divided into three parts, one to be kept unaltered, one to be reburned under the same conditions, and a third to have two reburns. In addition to this, pieces of quartz rock received one and two burns respectively, with the view of obtaining the same data.

The primary object was to determine, if possible, how far the inversion from quartz to cristobalite had progressed under the successive conditions. Each burn was made as nearly uniform as practicable, and in each succeeding burn the effort was made to maintain identical conditions. At that part of the kiln where these brick were burned the temperature reached 1,540° C. (2,804° F.), and they were subjected to that heat for approximately 40 hr.

* In general, the chief optical constant serving to distinguish cristobalite on the one hand from quartzite and calcium silicate on the other was the index of refraction. The slow ray of quartz has a refractive index of 1.544 while the index of the fast ray of cristobalite is about 1.485. The immersion method was used to determine the indices of the grains, the powdered brick being immersed in a liquid of index 1.51. By this method it was possible to distinguish between the higher-refracting quartz and calcium silicate and the lower-refracting cristobalite.

With the aid of a *camera lucida*, the fields were sketched on very thin tracing paper. The grains, as represented on the paper, were then cut out and weighed for each constituent. Aside from the fact that the composition of different fields may vary, other errors are entailed in the sketching, cutting out and weighing so that a variation of 5 per cent. is justifiable. In these determinations the fields of the same specimen

* The heartiest appreciation is here expressed to A. A. Klein, Petrographer in the U. S. Bureau of Standards at Pittsburgh, for his courtesy in making these determinations that they might be concluded in time for presentation here. The quality of Mr. Klein's work needs no comment.

checked to within 5 per cent. maximum and 1 per cent. minimum error.

Results.—The tabulated data are as follows:

Specimen	Field	Quartz and Calcium Silicate, Per Cent.	Average, Per Cent.	Cristobalite, Per Cent.	Average, Per Cent.
Lot 1.....	1	24.87		75.13	
(1st burn).....	2	20.42	22.65	79.58	77.35
Lot 2.....	1	14.31		85.69	
(2nd burn).....	2	19.96	17.13	80.04	82.87
Lot 3.....	1	17.44		82.56	
(3rd burn).....	2	14.60	16.02	85.40	83.98
Quartz rock.....	1	50.08		49.92	
(1st burn).....	2	51.05	50.57	48.95	48.95
Quartz rock.....	1	32.70		67.30	
(2nd burn).....	2	30.06	31.38	69.94	68.62

A number of interesting deductions may be made even from these preliminary tests. The maximum change takes place at the first burning, followed by a further appreciable inversion at the second, and practically no change at the third.

The grain structure of the reburned silica brick may throw some light on this subject. Here the grains may be seen to be essentially cristobalite, with the quartz occurring as an inclusion. The last stages of the inversion are apparently exceedingly slow within the protecting envelope.

The smaller percentage of inversion in the case of the quartz rock as compared with the brick is very noticeable, and is undoubtedly due for the most part to the grinding of the quartz in the manufacture of the brick. This is wholly in line with Fenner's laboratory experiments, in which he found the influence of comparative grinds to be slight if the silica mineral were heated with a flux, but to be marked if the material were heated without a flux. Fine grinding greatly hastened the inversion. The greater freedom of movement in expansion as compared with the exceedingly dense quartz body is probably the controlling factor, rather than any difference in the penetration of the heat.

The part played in the inversion (if any part is played) by the lime and its compounds present, is of much interest. It appears logical that the greater fluidity conferred by the fluxes present should have an influence; but it must be borne in mind that, as compared with previous experimental work on silica minerals, the amount of flux present in the brick is exceedingly small, and that definite chemical compounds are formed in combination with the brick body. In Fenner's work the sodic tungstate formed no such chemical compounds, and remained unchanged at the conclusion.

The fact moreover that no tridymite was determinable in either the burned brick or the calcined quartz is explained by the presence of the

negligible amount of flux. This would favor the direct inversion of quartz to cristobalite.

Thus far, we have assumed the changes in crystal form to be desirable, because of their effect on the expansion of the material.

Effect of Burning on Physical Strength

It is also of interest to note the effect of the reburning on the physical strength of the brick. To determine this, the following tests were run:

Cross-Breaking or Modulus of Rupture Test

Number of Burns	Mark	Breadth of Brick, Inches	Depth of Brick, Inches	Weight Causing Fracture of Brick, Pounds	Length between Supports, Inches	Modulus of Rupture, Pounds Per Square Inch	Average, Pounds
1	9 A	2.46	4.40	3,550	6	671	
1	9 A	2.42	4.40	3,000	6	576	624
2	9 B	2.50	4.50	4,140	6	730	
2	9 B	2.50	4.50	5,000	6	889	809
3	9 C	2.06	4.50	4,500	6	971	
3	9 C	1.96	4.50	4,550	6	1,031	1,001

Crushing-Test: Whole Brick Crushed Flat

Number of Burns	Mark	Dimensions, Inches	Area, Square Inches	Load, Pounds	Crushing Strength per Square Inch, Pounds	Average, Pounds
1	9 A 2	9.00 by 4.40 by 1.96	39.60	176,800	4,465	
1	9 A 2	9.04 by 4.44 by 2.04	40.14	167,000	4,161	4,313
2	9 B 2	9.10 by 4.46 by 2.02	40.58	186,500	4,596	
2	9 B 2	9.10 by 4.40 by 2.06	40.04	168,000	4,196	4,396
3	9 C 2	9.06 by 4.42 by 2.00	40.04	195,000	4,870	
3	9 C 2	9.12 by 4.46 by 2.06	40.68	168,000	4,130	4,500

The increased strength of the brick is evident.

CONCLUSIONS

In considering whether or not all the expansion possible is burned out of the brick, it is necessary to determine first the limits and then how closely they are approached.

Day, Sosman and Hostetter say³ that the true specific volume of cristobalite as compared with quartz at 20° C. is 13.4 per cent. greater.

³The Determination of Mineral and Rock Densities at High Temperatures. *American Journal of Science*, 4th series, vol. xxxvii, No. 217, pp. 1 to 39 (1914).

Assuming a green brick to be wholly pure quartz (100 per cent. SiO_2), and the brick burned from it wholly cristobalite, the volume increase would then be 13.4 per cent. if the relative porosity of the body be disregarded. This theoretically possible expansion is diminished by the effect of the impurities present in the actual brick, and by the resulting compounds formed. For instance, 4 per cent. of impurities would produce a much larger proportion of compounds to which this expansion phenomenon does not apply.

An expansion of, say, $\frac{3}{8}$ to $\frac{7}{16}$ in. per foot, to which byproduct coke-oven silica is burned, thus provides for somewhat more than 13 per cent. increase in volume.

It must be borne in mind, however, that by quick burning the brick may be easily expanded by double or treble the normal amount; but sound brick do not result. This excessive expansion is due to minute fissures opened up in the body of the brick by a too sudden rise in temperature during burning. In the early stages of firing the heat must be raised very slowly. In general the conclusion is justified that the present good commercial practice carries the results of the quartz-cristobalite inversion to the greatest degree economically possible and that the measure of the expansion of the brick affords a closely approximate measure of the inversion—providing only that the brick are sound.

The short series of tests outlined above are given with full realization that, while indicative, they are by no means conclusive. They were but the preliminary runs, necessarily made somewhat in haste, but giving data which permit intelligent plans to be made for the more comprehensive tests now under way. There is much to be learned as to the effect of both time and temperature as well as the influence of pressure, which may produce a shifting of some of the inversion points. Judging from the work thus far done, it is impossible to expect the full inversion of quartz to cristobalite under any practical operating conditions of manufacture.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, Sept., 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Mine Pumping

BY CHARLES LEGRAND,* DOUGLAS, ARIZ.

(San Francisco Meeting, September, 1915)

THE problem of mine pumping is so much affected by local conditions, and those conditions are so liable to changes during the life of a mine, that the best system to use is difficult to determine. The experience of the writer has been that, in general, for copper mines, electric pumping is most satisfactory unless the quantity of water to be pumped is great or the mine does not use electric power for any other purpose. As a rule, however, figuring on the cost of necessary power plant, the total cost of installation is greater with electric pumps than steam pumps.

As the water is liable to be gritty, outside-packed plunger pumps are to be recommended, and if the water is not acid, or only slightly so, chilled cast-iron plungers pay for their extra cost very quickly in diminished cost of packing. With chilled plungers it is possible to use metallic packing instead of soft packing, if the water is not too gritty nor the lift too high.

In vertical pumps this packing is satisfactory for lifts up to 400 ft., in the experience of the writer, and may be found satisfactory for higher lifts.

For water slightly acid, cement-lined pump bodies have given good satisfaction.

For acid water, both plungers and pump bodies should be made of acid-resisting bronze.

For high lifts, especially with gritty water, a satisfactory pump valve is difficult to find. To reduce the unbalanced pressure on the valve at time of opening it is advisable to have narrow seats; this, however, brings high pressures on the seats when the valve is closed. On large valves for lifts of 600 to 1,000 ft. we have used leather, hard rubber, vulcanized fiber, and soft brass; for clear water the latter has given best satisfaction.

The difficulties due to grit make it advisable to have large sumps where water can settle, and if possible, to have two of them so that they can be cleaned alternately.

At the mines of the Old Dominion Copper Mining & Smelting Co., Globe, it was found that a small air ejector discharging into a mine car is a very convenient way to clean mud and sand from a sump.

* Consulting Engineer, Phelps, Dodge & Co.

The steam pumps can be divided into two broad classes: The direct acting, with simple, compound, or triple-expansion steam cylinders; and fly-wheel pumps with compound or triple-expansion steam cylinders. All the steam pumps are usually run condensing, both for steam economy and because exhaust steam cannot be discharged in the mine.

The steam consumption per water horsepower diminishes in the order given above and the complexity of the pumps increases as the steam consumption diminishes.

The simple direct-acting pump is so uneconomical in steam consumption that it should not be used for permanent pumping although very simple and convenient in emergencies. It can also be used with compressed air instead of steam.

The compound direct-acting pump for small power is a satisfactory pump.

The triple-expansion direct-acting pump is fairly economical in the use of steam if proportions of steam and water ends are correct for the lift. In a great many installations this is not the case and the steam end of the pump is too large for the work done, increasing considerably the steam consumption per horsepower. It is quite usual for mine managers to specify a higher lift for the pump than they expect to have when the pump is installed, so as to enable them to put the pump at a lower level later on; if a pump has to be ordered for such a change of lift it will be found advantageous to order the pump for the correct lift but specify the water end strong enough to stand the maximum pressure expected in the future. When lift on pump is increased the size of plunger can be altered at small cost to keep the correct proportions between steam and water end. This naturally decreases the capacity of the pump in proportion to the lift.

All the direct-acting pumps take comparatively little room and are easy to move from level to level.

The compound and triple-expansion fly-wheel pumps are more economical in use of steam, but are much more expensive of installation, require more room, and must be installed on good foundations that will not move or move as a whole. Their use can generally only be justified if the quantity of power to be delivered is considerable, steam expensive, and the probable life of the mine will repay the difference in cost.

Whether a surface condenser or a jet condenser should be used depends on local conditions. The surface condenser has less moving machinery and uses less steam than the jet condenser, but is liable to be out of service a longer time while tubes are being cleaned.

The question of proper size of steam line between boilers and pumps is one which usually is not given sufficient attention, most steam lines being too large for maximum economy. The writer has found in most cases that it is advantageous to have a steam pipe which gives a drop of pressure of 5 to 10 lb. between boilers and pumps, as the reduced con-

densation pays for the loss of pressure. When a small pipe line is used the usual separator at the pump should be made a separator-receiver, to steady the flow through the pipe line.

All steam pumps have a great advantage where flow is variable, as the speed and delivery of pump are easily adjustable to varying conditions.

The electric pumps can be divided in two broad classes: The plunger and centrifugal pumps.

The plunger pumps are of many designs; the writer generally prefers the vertical type as giving more even wear on packing and plungers and taking less floor space, although they require more head room.

For high lift the quintuplex pump having a practically steady flow on the discharge is advantageous, as it can be used without air chamber.

The motors are generally geared to the pump through a single or double reduction of gears. If straight spur gears are used it is not advisable to have a single reduction of more than 8 to 1, and slow-speed motors have to be used. If herringbone gears are used single reduction of larger ratio is permissible, allowing a motor of standard speed for pump drive. The herringbone gear is more efficient than the spur gear, but requires much closer adjustment of pitch line, so that bearings have to be made adjustable to maintain proper distance between pinion and gear shafts. For this style of drive the proper design of bearings is very important.

A flexible coupling between pinion and motor shaft usually pays for itself by reduced maintenance of motor, although on pumps using 50 hp. or less, a motor with outboard bearing, and either a fabric or raw-hide pinion geared directly to pump without coupling, has been found quite satisfactory.

With alternating-current motors, unless special motors are used, it is necessary to unload the pump at start by means of a bypass valve. Motors with high starting torque can be obtained, but are less efficient than standard motors, and are not necessary unless the pump is to be used with an automatic starter.

Direct-current motors will start the pump under full head, with a starting rheostat made for this service.

The writer has had little experience with centrifugal pumps. As a rule, however, they have the advantages of great simplicity, and ease of connection to motor, having no gearing; they are, however, less efficient than plunger pumps and have to be designed for exactly known conditions. They are sensitive to change of motor speed or changes of head, in a high-lift pump of moderate capacity the passages are small, the water travels at high speed, and if at all gritty the wear upsets the proportions of the pump and quickly affects the efficiency.

With plunger pumps the amount of water delivered can only be altered by varying the speed of the motor, starting and stopping the pump

as required by the flow of water, or by passing a portion of the water to the pump suction.

The first method is easy where direct current is available, but more complicated or less efficient where alternating-current motors are used.

The second method can be used with both systems of power, but is hard on the power plant if pumping load is a large proportion of the total.

The third one is inefficient in the use of power, as the pump is working at full load at all times.

As the efficiency of electric pumps is not much affected by their size, where flow of water is variable it is advisable to have more units of a smaller capacity so that the number of units running can be adjusted approximately to the quantity of water pumped and one unit started and stopped by hand or automatically to take care of the variation of water level in the sump.

Electric pumps usually require less attendance than steam pumps and reduce the quantity of heat liberated in the mine, which is a considerable advantage in hot mines.

The electrical measurements are so much easier to make than the steam measurements that an electric pump is likely to be kept at a higher efficiency. With an electric pump it is easy to show the attendants the evil effects of packing the glands too tight and the large amount of power that can be thus consumed without heating the plungers.

The electric plunger pumps can be operated at a lower lift than they are built for with very little loss of efficiency.

Where the proper submergence can be obtained an air lift is inexpensive to install and will handle very great quantities of water in a small space, and although the cost of the air is comparatively great, there are practically no other running costs. This method cannot be used from the lowest level of a mine, without a lot of complications, as there is then no way to get the proper submergence.

Air lifts were used at the mine of the Old Dominion Copper Mining & Smelting Co. at Globe in a recent emergency when water flow increased on the upper levels of the mine and got beyond the pumping capacity of the plant. A 10-in. air lift raising the water 200 ft. (exclusive of friction), using air at 90 to 95 lb. pressure at the power plant, required the following amount of air (measured by flow meter) at a barometric pressure of about 27 in. of mercury.

Gallons Water per Minute	Cubic Feet Free Air per Minute	Cubic Feet Free Air per 1,000 Gal.
1,011	1,353	1,338
1,680	1,809	1,080
1,794	2,262	1,261
1,925	2,658	1,375
1,965	3,219	1,638

Another lift raising the water 431 ft. exclusive of friction gave the following results:

Gallons Water per Minute	Cubic Feet Free Air per Minute	Cubic Feet Free Air per 1,000 Gal.
1,122	3,051	2,718
1,233	3,306	2,681
1,233	3,484	2,825
1,291	3,832	2,968
1,291	3,919	3,035
1,325	4,089	3,086

In these tests, which were kept up from 1 to 2 hr. each, the measurements of air were taken with a General Electric Co. air-flow recording meter installed carefully and checked after test according to instructions furnished by the makers; the results are probably correct within 5 per cent.

The measurements on water were obtained by taking the reduction of speed of the pumps when air lift was working and the water level in sump kept constant and a volumetric efficiency of 90 per cent. assumed for the pumps. All pumps were equipped with revolution counters.

The average speed of the pumps was recorded before and after the air test to insure that the water flow had not changed in the mine. The quantities of water are very nearly correct.

The above figures are not given as absolute, but only to give a rough idea of what can be done with an air lift. In both cases the submergence was about 190 ft.

In the following table are given the results of some tests on various sizes of steam and electric pumps. These tests were taken under running conditions and for steam pumps the water fed to boilers was taken as used by the pumps, all the auxiliaries of the boiler plant being run from another source of supply.

Tests 1, 2, and 3 were taken by condensing the exhaust of the pump and weighing the condensate and estimating the volumetric efficiency of water end at 90 per cent., as the delivered water was not measured.

Tests 4 to 9 were not taken by the writer.

Tests 10 and 11 are on triple-expansion four-cylinder pumps with two low-pressure cylinders. On these tests steam was superheated 35° when leaving boilers and the pumps were tested together.

Tests 12, 13, and 14 were on pumps immediately after starting.

Test 15 on same pump as No. 14 after two weeks' run.

Test 16 is on single-reduction spur-gear pump.

Test 17 is on single-reduction herringbone-gear pump.

Test 18 is the best test on air lift for 200 ft.

Test 19 is the next to the best test on same.

Test 20 is the best test on air lift for 431 ft.

Test 21 is the next to the best test on air lift for 431 ft.

On Tests 18 to 21 the steam consumption per water horsepower-hour is based on air compressors requiring 38 lb. of steam per 1,000 cu. ft. of free air, delivered compressed to 9 lb. gauge pressure.

Steam Pumps

Test Number	Kind of Pump	Steam Cylinders, Inches	Water Plunger, Inches	Stroke, Inches	Speed, Rev. per Min.	Water Del'd During Test, Gal. per Min.	Lift Including Friction, Feet	Steam per Water Horsepower-hour	Boiler Pressure, Pounds
1	Single Duplex.....	9	8½	10	33.0	270	120	98.0	135
2	Single Duplex.....	14	8½	10	33.0	285	111	135.0	135
3	No. 1 Exhausting in No. 2 as a Compound pump.....								
4	Triple Expansion Duplex... 10-15½-26		8	24	14.1	350	120	78.0	135
5	Triple Expansion Duplex... 15-23-39		10	24	12.7	278	591	61.5	122
6	Triple Expansion Duplex... 15-23-39		10	24	21.7	391	926	49.4	125
7	Triple Expansion Duplex... 18-28-47		12	24	13.0	658	629	42.1	131
8	Triple Expansion Duplex... 18-28-47		12	24	25.7	584	1,056	35.9	143
9	Cross-Compound Fly Wheel 19-35		5-½	36	32.4	1,207	1,056	27.8	142
10	Triple Expansion Fly Wheel 14-26-26-26		7-¾	36	38.0	502	1,059	23.7	110
11	Triple Expansion Fly Wheel 14-26-26-26		6-¾	36	32.0	878	652	18.0	137
				36		553	883	18.0	137

Electric Pumps

	Kind of Pump	Motor, Horsepower	Water Plunger, Inches	Stroke, Inches	Speed, Rev. per Min.	Water Del'd During Test, Gal. per Min.	Lift Including Friction, Feet	Steam Based on 25 lb. per Kw-Hr.
12	Vertical Triplex Double Re- duction.....	10	4	9	35	51	475	32.7
13	Vertical Triplex Double Re- duction.....	20	4	9	37	54	1,162	28.1
14	Vertical Triplex Double Re- duction.....	50	9	10	37	302	423	29.1
15	Vertical Triplex Double Re- duction.....	50	9	10	37	303	423	25.6
16	Vertical Quintuplex Single Reduction.....	110	4½	12	40	159	1,155	23.1
17	Horizontal Quintuplex Single Reduction.....	250	9	18	52	1,245	605	22.2
18				<i>Air Lifts</i>			Excluding Friction	
19						1,680	200	48.7
20						1,794	200	57.5
20						1,122	431	57.0
21						1,233	431	57.6

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Conveyor-Belt Calculating Chart

BY J. D. MOONEY AND D. L. DARNELL,* AKRON, OHIO

(San Francisco Meeting, September, 1915)

THE accompanying chart has been drawn for the convenience of engineers as a means of quickly determining the correct number of plies of conveyor belts operating under specific conditions.

The calculations are based on the average safe strength (factor of safety, 15) of the various standard rubber conveyor belts.

The calculations assume maximum loading conditions; that is, the belt is considered as carrying the greatest load that it will handle without spillage at ordinary belt speeds. This not only produces the most economical operating conditions, but also the maximum tension in the belt.

The chart is a graphical representation of the formula:

$$p = kgW(L + 10H)$$

Where,

p = the correct number of plies

k = a constant, depending on the type of drive

g = the weight in pounds per cubic foot of material handled

W = the width of the belt in inches

L = the length of the belt in feet (approximately twice the center distance).

H = the difference in elevation between the head and tail pulleys, in feet.

For a simple drive, with a bare pulley, $k = \frac{1}{250,000}$

For a simple drive, with a rubber-lagged pulley, $k = \frac{1}{300,000}$

For a tandem drive, with bare pulleys, $k = \frac{1}{375,000}$

For a tandem drive, with rubber-lagged pulleys, $k = \frac{1}{455,000}$

The chart is drawn for a simple drive, with a bare pulley ($k = \frac{1}{250,000}$);

therefore, the number of plies obtained from the chart should be multiplied by the factor 0.83 or $\frac{5}{6}$ for simple, lagged drive; the factor 0.67 or $\frac{2}{3}$ for tandem, bare; and the factor 0.55 or $1\frac{1}{20}$, for tandem, lagged.

The formula $p = kgW(L + 10H)$ has been developed mathematically

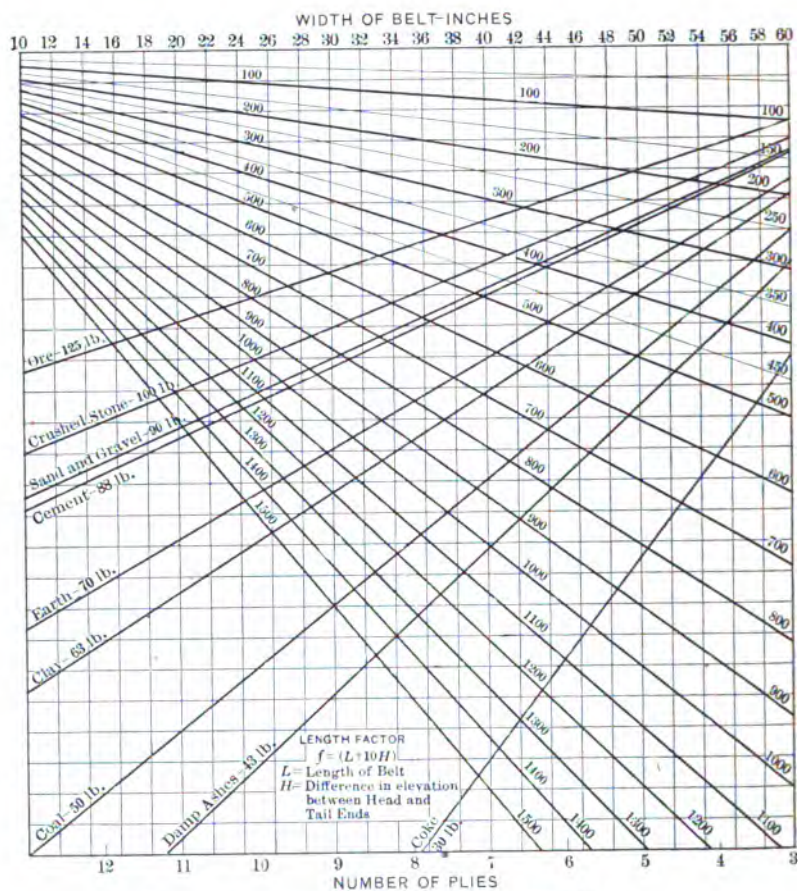


FIG. 1.—CONVEYOR BELT CALCULATING CHART.

from the following formulas, which have been found to work very satisfactorily in average good practice:

$$U = \frac{0.08W^2Sg}{5000}$$

$$HP = \left(\frac{0.02l}{100} + \frac{0.01H}{10} \right) U$$

$$T = \frac{C \times HP \times 100}{S}$$

$$p = \frac{T}{24W}$$

Where, U = Capacity in tons per hour
 W = Width of belt in inches
 S = Belt speed in feet per minute
 g = Weight per cubic foot of material handled
 HP = Horsepower developed in driving conveyor belt
 l = Length of the conveyor, in feet (approximately $\frac{1}{2}L$)
 H = The difference in elevation between the head and tail pulleys, in feet
 T = The total tension in the belt, in pounds
 p = Correct number of plies
 C = The constant of the drive.

For a simple drive, bare pulley,	$C = 600$
For a simple drive, rubber-lagged pulley,	$C = 500$
For a tandem drive, bare pulleys,	$C = 400$
For a tandem drive, rubber-lagged pulleys,	$C = 330$

The length factor, $f = (L + 10H)$, represented on the chart by the lines 500, 600, 700, etc., is a developed factor equal to the sum of the length of the belt and ten times the difference in elevation between the head and tail pulleys.

To find the correct number of plies for a conveyor belt, knowing the width, the length, the difference in elevation between the head and tail ends, and the kind of material to be handled:

Start from the width given at the top of the chart and move down until this line intersects the line corresponding to the proper length factor; then move either right or left until the line corresponding to the given material is met; then move down again to the scale of plies, where the next largest figure will give the correct number of plies.

For example: To find the correct number of plies for a conveyor belt 36 in. wide and 300 ft. long, with 20 ft. difference in elevation; handling sand and gravel:

Follow the line from the 36-in. width downward until it intersects the 500 length factor line; then follow to the right until the sand and gravel line is intersected; then down to the ply scale, where the ply will be found to be 7.



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Ventilation of the Copper Queen Mine

BY CHARLES A. MITKE,* BISBEE, ARIZ.

(San Francisco Meeting, September, 1915)

INTRODUCTION

THE Copper Queen mine is composed of seven divisions which are operated through the following shafts:

Division No.	Shaft	Depth, Feet	Air Current
1	Uncle Sam	600	Downcast (shut down April, 1914, to curtail production)
	Southwest	600	Downcast
2	Czar	400	Downcast
3	Holbrook	600	Downcast
4	Spray	800	Downcast (shut down April, 1914, to curtail production)
5	Gardner	1,000	Downcast
6	Dallas	1,400	Downcast
	Lowell	1,600	Upcast
7	Sacramento	1,700	Downcast

The workings of the different shafts are connected by motor-haulage drifts on the even numbered levels. The general location of the ore-bodies and workings is illustrated by the vertical projection of orebodies of the district as shown in Fig. 1.

The Uncle Sam, Southwest, and Czar workings have many connections to the surface through raises and extensive cracks that were caused by moving ground. These divisions are cool and are ventilated entirely by natural means.

The Holbrook is ventilated partly by natural and partly by artificial ventilation, while the Spray, Gardner, Lowell, and Sacramento are ventilated entirely by mechanical means.

In some of the divisions mentioned above a considerable quantity of air exhausts through shafts of adjoining properties.

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CONDITIONS UNDER NATURAL VENTILATION

Several Mine Fires Occurred

During the last few years several mine fires menaced both life and property. The most serious mine fire was that of the Lowell, which occurred during 1911. The district where the fire originated was between the 1,000 and 1,200 ft. levels. The orebody contained a high percentage of sulphur, and the gob a large amount of pyrite; as stoping progressed the friction between the blocks of sulphides increased, due to moving ground. Oxidation of pyrite and timber was also an important factor. The temperature increased until the timbers took fire, which in turn started fire in the sulphide ore. The gases resulting from this fire, such as SO_2 , and distillation products of wood, came through the upcast shaft and caused considerable damage, thus putting the shaft out of commission. By reason of broken ground, the fire could not be sealed off at that time, which made necessary the repair of 800 ft. of the shaft by men using oxygen helmets.

Several fires also occurred at different times in some of the other divisions. They were particularly dangerous under conditions of natural ventilation as gases and smoke usually entered the workings. One of the reasons, therefore, in planning a system of mechanical ventilation was to provide a better means of control immediately after a fire started.

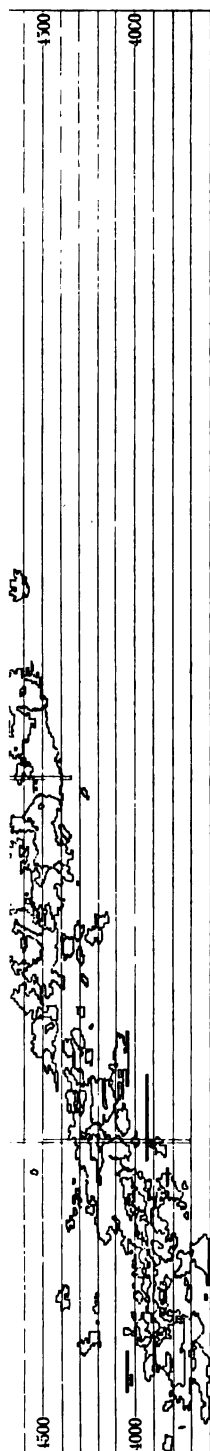


FIG. 1.—VERTICAL PROJECTION OF OREBODIES. Plane of projection bears S. 32° $46'$ $50''$ E.

General Working Conditions Investigated

During 1912, the company decided to make an investigation of the general working conditions of the Gardner division. By reason of the scarcity of connections with other mines this division contained the poorest working conditions and presented the greatest problem for ventilation. Readings were taken on natural air currents as to their velocity, volume, direction, temperature, relative humidity, CO₂ content, etc. This indicated that even though the entire mine seemed to be hot the average temperature was not high, varying from 75 to 100° F. The relative humidity, however, was exceptionally high everywhere in the mine, ranging from 90 to 100 per cent.

The CO₂ was under 1 per cent. in the general run of working places and from 1 to 3.5 per cent. in exceptionally remote workings. The velocity and volume of the air currents were very low. Moisture was deposited on timbers and the sides of drifts, making the mine damp; a species of fungi covered a large percentage of the timbers. About 225 men worked in this division on each shift, and from the standpoint of efficiency the quantity of fresh air per man per minute was not all that was desired. Men in hot stopes had to be changed every two or three days on account of the heat, and in this way by the time a man was familiar with the workings of his stope he was put into another place. Consequently, all the men who had fairly good working conditions had to share the ordeal from time to time, by working in these undesirable places.

It was the custom when conditions became particularly bad to drive new air connections. In a few instances they proved beneficial, but in the large majority of cases the quantity of air which passed one way or the other was so small that the effect was negligible.

THE RELATIVE MERITS OF PRESSURE AND EXHAUST SYSTEMS CONSIDERED

At first, plans were formulated and data obtained, as described under *Planning the System of Mechanical Ventilation*, for both the pressure and exhaust systems. After the merits of each were carefully considered, the pressure system of ventilation was adopted. According to this system, all mine fans were placed underground on the deepest working levels of the different divisions so as to draw the air down the intake shafts, force it through the workings and stopes, and exhaust it to the surface through upcast shafts.

Four Reasons for Adopting Pressure System

The reasons for adopting the pressure system instead of the exhaust are as follows:

1. *More Practical.*—Since all the deep shafts of the Copper Queen are in operation and a large amount of work is done around each, as for example, men going on and coming off shift, handling timber, tools, and supplies, etc., it would not be practical to have an exhaust fan on the surface near the collar of the shaft.

In the Gardner, a subway below the collar of the shaft would be out of the question. Consequently, a large air-tight chamber, inclosing nearly the entire head frame, would have to be built if a suction fan were used. This arrangement would be expensive and unsatisfactory.

2. *Better Control of Mine Fires.*—The pressure system is the more advantageous in case of mine fires. With this arrangement, the entire workings are under pressure and the gas or smoke from a mine fire may be coursed to exit through neighboring shafts that are not in operation. Since the air is under considerable pressure at the bottom and sides of the present fire districts and each of the outlets over them is provided with regulator doors, the quantity of air passing through the fire zone can be increased or diminished at will and the fire consequently can be brought under control more readily than with the exhaust system.

3. *Greater Efficiency.*—Since the Spray, Gardner, Dallas, and Sacramento shafts are natural downcasts, the mine fans were placed underground so as to take advantage of these circumstances and multiply the volume of the natural downcast from five to ten times. The pressure system, therefore, has the advantage of the additional natural ventilation, which evidently would be lost with the other system; and more power would be required for the same quantity of air if the natural downcasts were changed to upcasts by suction fans.

4. *Greater Safety.*—Experience in the Lowell mine fire showed that in the event of the connections being closed between the fire and the suction fan, there is a possibility of smoke and gases entering the workings below and endangering the men. In the adopted system, the maximum air pressure is below the fire district, which makes it impossible for gases to go down, even if bulkheads of a sealed fire district should be opened suddenly by caving ground.

At present a churn-drill hole is being put down to a depth of 1,100 ft. over the old Lowell fire stopes. When this is completed water will be run in to cool the district. No trouble is anticipated since this was thoroughly tried at the Holbrook, where a large amount of water was turned into churn-drill holes and no smoke or gases were taken down against the pressure of the mine fan.

EQUIPMENT AND GENERAL ARRANGEMENT FOR MECHANICAL VENTILATION

The following pages give an outline of the general mechanical ventilating system which was adopted at the Copper Queen mine.

Combined Natural and Mechanical System in Holbrook and Gardner Divisions

Because of the many openings to the surface, the natural ventilation is satisfactory from the surface to the 400-ft. level in the Holbrook division. Below this level mechanical ventilation is used, as indicated in Fig. 2. There is a 20-in. blower on the 600-ft. level, which delivers 8,000 cu. ft. of air per minute. Since very little stoping is done between the 400- and

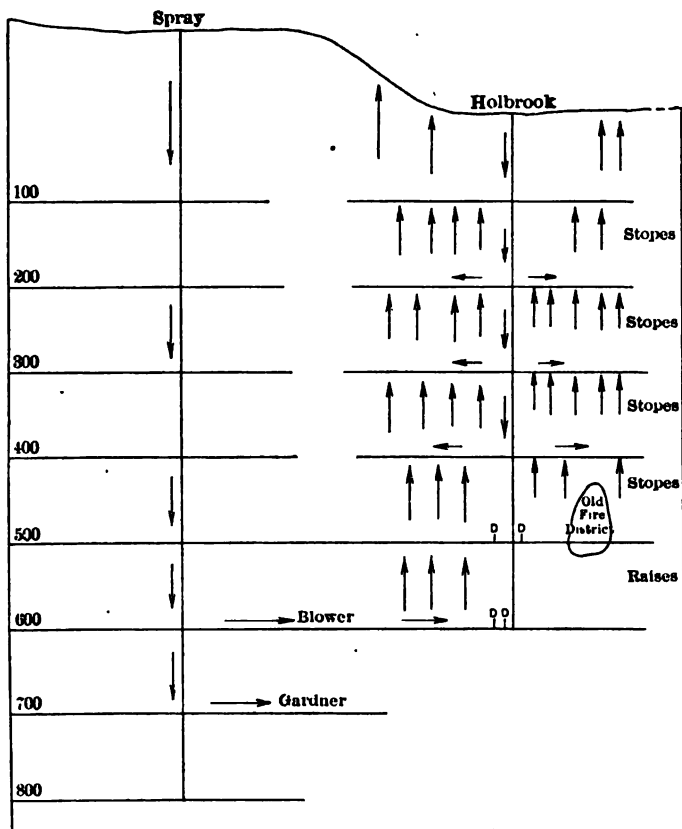
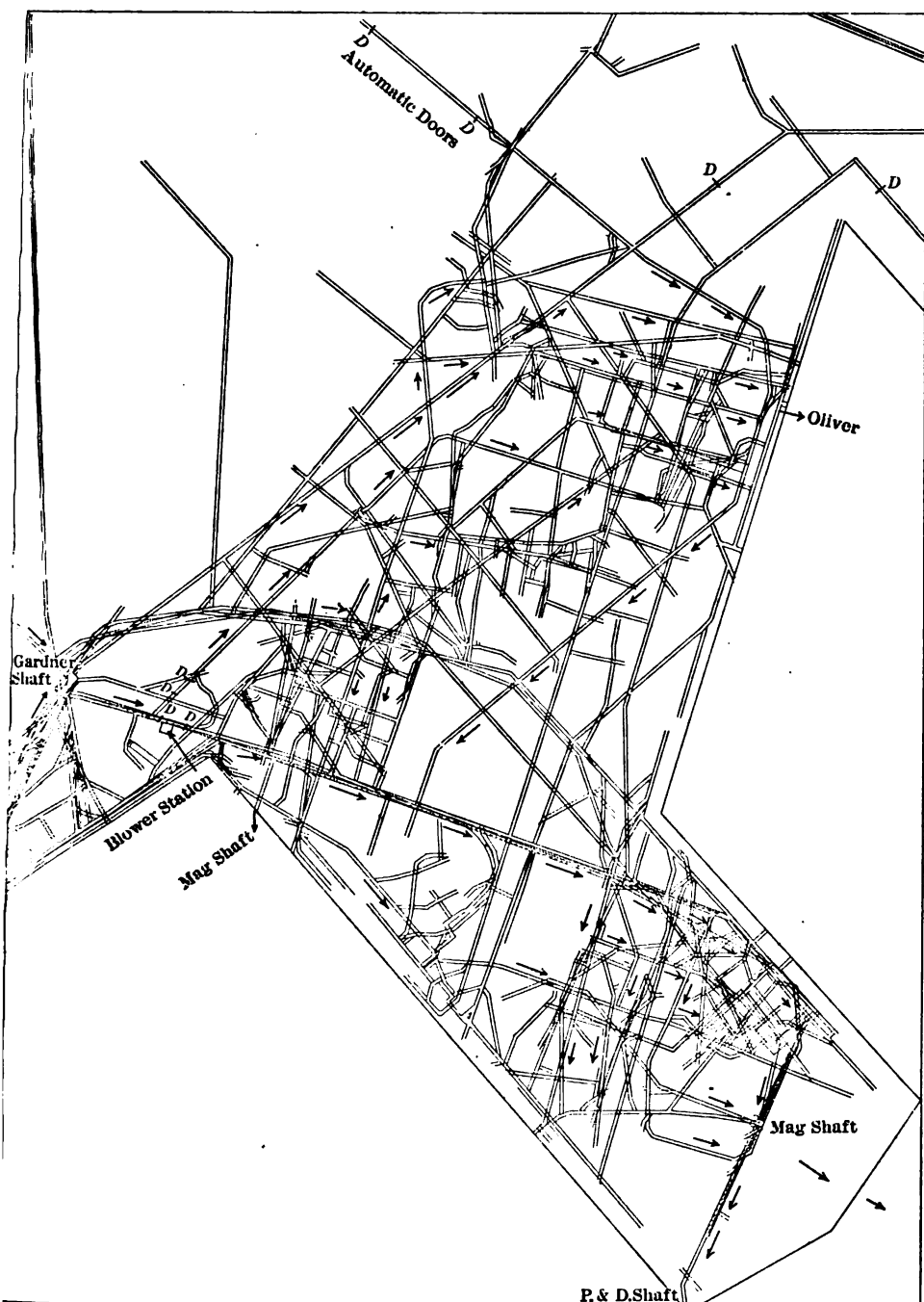


FIG. 2.—NATURAL AND MECHANICAL VENTILATION OF HOLBROOK DIVISION.

600-ft. levels, the blower is operated principally to keep a pressure around the old fire district to prevent any gases which may escape from entering the workings.

The outlines of the natural and mechanical ventilating system of the Gardner division are shown in Figs. 3 and 4. The arrows indicate the direction of some air courses and DD represents a few double doors. On the 900-ft. level, there is a Sturtevant mine fan delivering 72,000 cu.



• FIG. 5.—ARROWS INDICATE DIRECTION OF SOME IMPORTANT AIR CURRENTS.

turn and at the same time allow the door to be opened. The cost of upkeep is small and the entire equipment only requires the amount of attention usually given to any machine.

The automatic doors are used on motor tracks only, while small hand doors are used in all ordinary drifts. The standard hand door is 6 ft. 6 in. high, 3 ft. 9 in. wide and $2\frac{1}{4}$ in. thick, made up of 1-in. boards with paper between the two thicknesses. The posts of the door frame are 6 by 10 in. by 6 ft. 6 in. and the cap 6 by 10 in. by 4 ft. 3 in. The motor

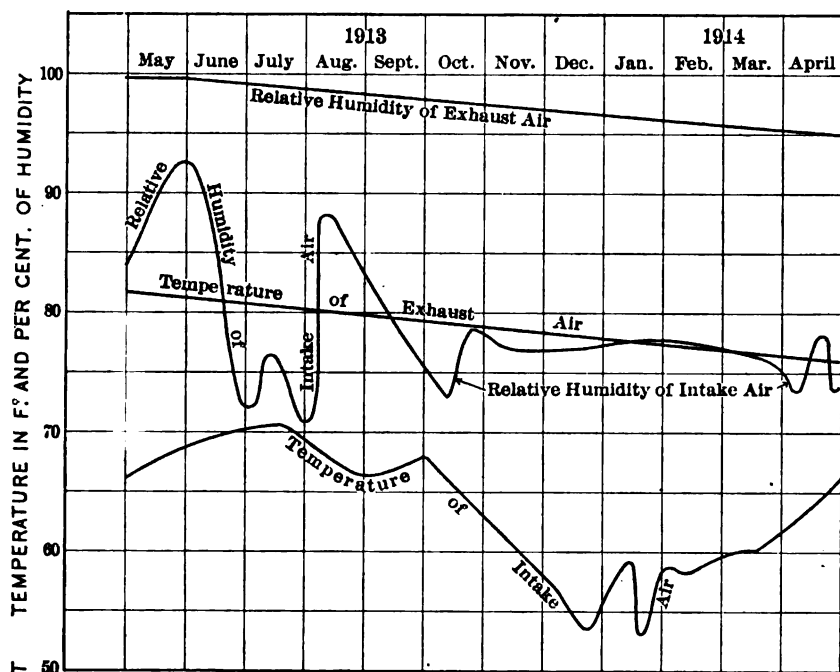


FIG. 7.—CHART OF TEMPERATURE AND RELATIVE HUMIDITY OF INTAKE AND EXHAUST AIR BEFORE AND AFTER NEW VENTILATION SYSTEM WAS IN OPERATION.

tracks are mounted on a timber sill set in concrete. The space between the bottom of the door and the sill is filled with concrete, except for clearance spaces for the flanges of the car wheels. The spaces between the door frame and the walls and roof are likewise filled with concrete. The automatic door is 6 ft. 11 in. high by 4 ft. 4 in. wide made of 1 by 12 in. boards faced on both sides with $\frac{3}{4}$ by 2 in. flooring.

Air Jets Used in Hot Workings.—Formerly small electric blowers were used to ventilate hot raises and stopes. These poorly ventilated places were usually some distance from the shaft, which required a considerable investment and long electric cables. Instead of using these blowers an air jet was designed and tried for workings of this type. The jet had

eight $\frac{1}{16}$ -in. holes and was placed in the center of a 14-in. funnel which was connected to a 10-in. pipe. About 800 cu. ft. per minute was delivered at the end of 50 ft. of pipe of this kind. The cost of compressed air is comparatively small and is less than one-third the cost of electrical power, when used in these particular instances. Of course the jet is only used until the connection is made. After connections are made considerable quantities of air pass through these drifts and raises from the large blower, and the ventilation is generally satisfactory.

New System Lowered Temperature and Relative Humidity

The chart, Fig. 7, illustrates the decrease in temperature and relative humidity of the intake and return air before and after the blowers were started. On account of the large amount of moisture in the ground and also the wet conditions of moist drifts and workings, it took about four months to dry the timbers and working places after the new ventilating system was in operation.

Velocity and Volume of Air Increased

The blowers were started June 1, 1913, resulting in an increase in velocity of the intake air from 65 ft. per minute on June 1, to 1,000 on June 30 and 1,160 on July 31. The volume also increased from 10,000 cu. ft. per minute on June 1, to over 60,000 on June 30 and 70,000 on July 31. The reduction in the temperature and humidity was not so marked as the increase in velocity and volume. The division in general, however, seems much cooler and the environment is comfortable.

Lowell and Sacramento Divisions Ventilated Jointly

The natural and mechanical ventilation systems of these divisions are outlined in Figs. 8 and 9. There is a Sirocco blower (capacity 35,000 cu. ft.) on the 1,400-ft. level of the Dallas shaft, forcing the air towards the Lowell against a $1\frac{3}{8}$ -in. water pressure and another blower on the 1,600-ft. Sacramento (capacity 45,000 cu. ft.) operating against a $1\frac{1}{2}$ -in. water gauge. The reason for the higher pressure in the Sacramento is to have the air from that level join the Dallas air on the 1,400-ft. level and from there rise through the workings to the 900-ft. level of the Lowell, and exhaust through the two shafts as indicated in Fig. 9.

Part of Fire District being Cooled.—An example of the effect of large volumes of air passing through a district, part of which is on fire, may be obtained from the 13-10-10 stope. Here the temperature has been reduced from 87 to 83° F. To accomplish this result, 15,000 cu. ft. of air per minute was driven through this stope from Sept. 1, 1914, to Feb.

1, 1915, a period of five months. Before this blower was started the humidity was about 98 per cent. and the velocity of the air was scarcely noticeable. After the 15,000 cu. ft. of air per minute was driven through the stope, even though the temperature showed a very little difference, the humidity dropped to about 80 per cent. and the velocity increased to about 400 ft. per minute. As a result the stope seems cool and the men can do efficient mining. This small reduction in temperature also shows that it requires a large quantity of air circulating a long time in a fire district to cool the country rock.

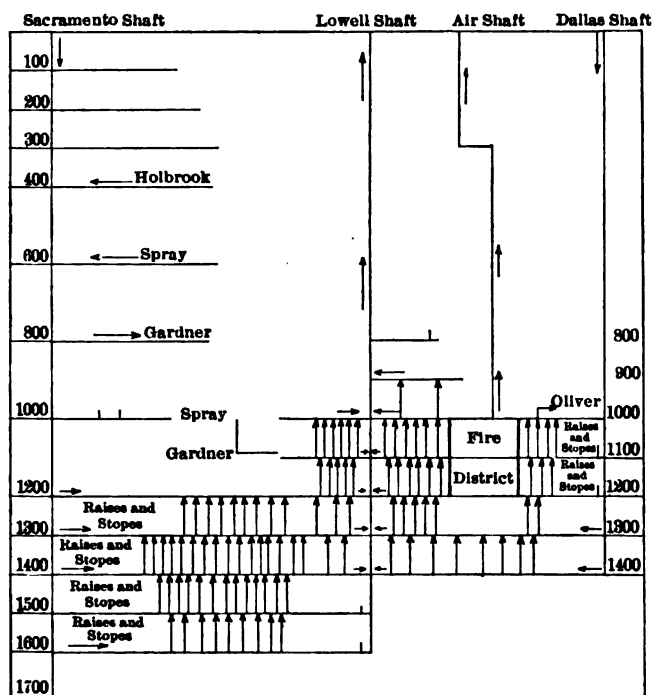


FIG. 8.—OUTLINE OF NATURAL VENTILATION SYSTEM OF LOWELL AND SACRAMENTO MINES.

The velocity and volume of air have been increased through all the stopes and workings and although the temperature throughout the entire mine has not been lowered much, nevertheless the workings seem much cooler on account of the increased velocity and volume and decrease in humidity.

DATA COLLECTED BEFORE MECHANICAL VENTILATION SYSTEM PLANNED

In order to plan the general system of mechanical ventilation of the Copper Queen, as outlined above, the following methods of obtaining

data for a working basis were pursued. It may also be stated here that these methods would be the same for either pressure or exhaust systems.

Instruments for Determinations.—The following instruments were used in determining the temperature, relative humidity, velocity and volume of mine air: The anemometer, water gauge, hygrometer, and barometer.

Recording of Data.—Data were readily obtained in the form of daily reports.

Barometric readings, when taken occasionally in important places, were satisfactory for all practical purposes, while readings with the anemometer and hygrometer were made daily for several months.

The following is an example of the daily report blank used at the Copper Queen:

Ventilation Report

Shaft	Level	Dry Bulb	Wet Bulb	Humidity	Bar	Date					
Location	Date	Time	Area	Veloc.	Cu. Ft.	Bar	Dry Bulb	Wet Bulb	Relative Humidity	CO ₂	Air Current
Average											

Two Standards of Measurement

There are two standards by which the degree of ventilation in a mine may be measured: (a) a quantity standard—that is, the quantity of pure air entering the mine per man, per minute; or, (b) the quality standard—this being determined by the amount of impurity present. These standards are entirely different and each one has its particular application.

The quantity standard is less expensive, more practicable, and is the method most generally used in both coal and metal mines. The quantity

standard was used in all drifts, raises, and places where there are air currents that can be detected and measured with the anemometer. This method is also the more important because the increase of velocity of air materially affects the efficiency of the men.

The quality standard is very expensive and its application is only necessary in exceptional cases. It has only a remote bearing on the efficiency of the men, but occasionally it is found advisable to have accurate determinations made on several samples of mine air.

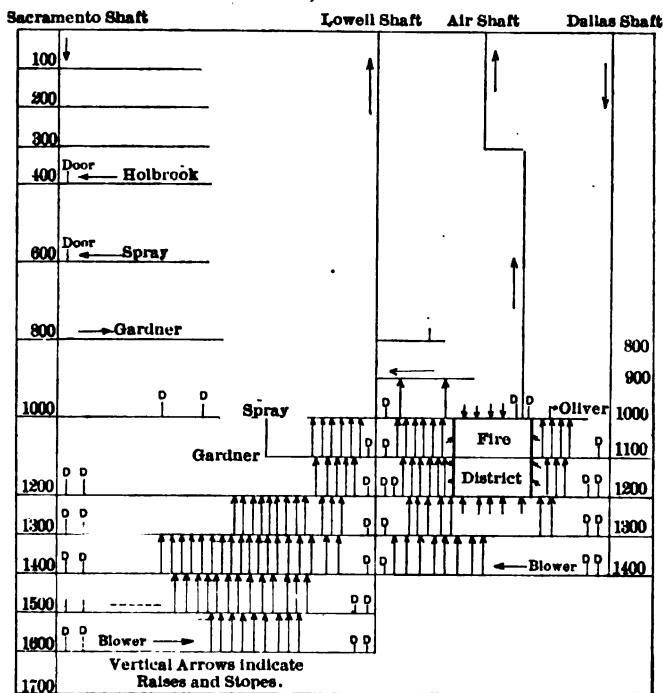


FIG. 9.—OUTLINE OF MECHANICAL VENTILATION SYSTEM OF LOWELL AND SACRAMENTO MINES.

Averages Plotted on the Mine Map

The data secured as explained in the paragraph on "Recording of Data" were averaged every week. Blueprints of the different levels of the mine were made and the monthly averages written on them in their proper places. The direction of air currents was also indicated by arrows, especially where readings were made with the anemometer.

Amount of Air Necessary for Each District Calculated

In order to calculate the amount of air necessary for each district, it was necessary to take into account the following:

1. Number of men and animals in each district.
2. The production of CO_2 , or other gases, as shown by the chemical analyses.
3. Relative humidity.
4. Temperature.
5. Amount of explosives used.
6. The distance from currents of good air.
7. The number of lights.
8. Air leakage.
9. Friction of the air currents.
10. Number of splits of the air current.
11. Method of distribution.

Natural Ventilation Utilized Where Possible

In order to get the most economical system, it was necessary to utilize all possible natural ventilation and secure the proper distribution of the air. The latter is extremely important because the efficiency of the ventilating system depends largely on the proper distribution of the air currents. The arrangement of air currents by mechanical means was designed to coincide with that of natural ventilation in nearly all cases.

Natural Ventilation Supplemented by Mechanical Means

In the divisions where natural ventilation was not sufficient, the natural air currents were strengthened by mechanical, or artificial means, by installing large mine fans.

A Good Working Atmosphere Dependent on Several Factors

There is a diversity of opinion among authorities on what constitutes "a good working atmosphere," and to what extent it will be economical to ventilate a mine, considering the cost of installation, maintenance, and power. The quantity of air necessary for the ventilation of one mine would be inadequate for the ventilation of another. The same can be said regarding the quantity of air per man per minute, consequently every mine is an individual problem, and while generalities are made, to get the greatest efficiency of a ventilating system, a detailed study of the mine must first be made, and the new ventilating system adapted to meet the particular needs of each mine.

In a number of States the quantity of air in coal mines required by law is usually specified, as, for example: The Anthracite Mine Law of Pennsylvania specifies a minimum quantity of 200 cu. ft. per man, per minute. One of the modifying clauses of the above law stipulates that the amount of air in circulation should be sufficient "to dilute, render harmless, and sweep away, smoke and noxious or dangerous gases."

Even in coal mines, therefore, where the problems of ventilation have not the intricacies found in metal mines, there is no fixed standard upon which to base figures for "a good working atmosphere" or an efficient ventilating system.

Experience in ventilation of the Copper Queen mine has shown thus far that in poorly ventilated areas, comfort, efficiency, and economy are obtained by increasing the natural ventilation with mechanical means until the following conditions are obtained: Temperature, 77° F.; relative humidity, under 85 per cent.; velocity of air in working places at least 100 ft. per min.; volume of air in working places at least 250 cu. ft. per man, per minute. This may be termed a good working atmosphere and is the present standard for this camp.

In general, if the above velocity and volume are maintained in any working place it will not be necessary to make chemical analyses, because the percentages of CO₂, CO, NO₂, and SO₂ gases, etc., as well as the temperature and humidity will be low and require no attention, except in a few instances.

Increased Velocity of Air Improves Working Conditions

Formerly there were several examples of stopes in the Copper Queen mine which were similar to the following:

In No. 8-13-5 stope of the Gardner division, the readings in working places for May 28, 1913, were

Velocity.....	10 ft. per minute (approx.)
Volume.....	10 cu. ft. per min. (approx.)
Temperature.....	84° F.
Humidity.....	96 per cent.

A small 5-h.p. blower was forcing air into the stope. The miners were uncomfortable in this stope and complained about the excessive heat.

On Oct. 1, 1913, after the new ventilating system was installed, the following readings were made in the same stope:

Velocity.....	100 ft. per minute
Volume.....	500 cu. ft. per min.
Temperature.....	84° F.
Humidity.....	90 per cent.

The same miners mentioned above happened to be working in this stope on Oct. 1, 1913, and they remarked "the stope is not near as hot as it used to be and we can work much better." The men did not sweat excessively in this atmosphere, and the greater comfort experienced by them was due to the *increase in velocity*, which also means an increase in volume, with lower humidity, and not to the temperature, as it was the same in both cases.

When miners work in hot mines, drifts, and stopes, they generally neglect their personal safety. It was a frequent occurrence for the shift boss to enter a working place and after asking miners to bar down the back to discover that they had been working under loose boulders for several hours. It is evident, therefore, and proved by the rate of accidents, that men are on the alert and look out for personal safety far more in a comfortable atmosphere than in hot, damp, and disagreeable places.

In regard to efficiency, the tonnage per man shift has increased with the increase in the velocity and volume of air resulting from the installation of large mine fans. The greater efficiency of the men may be partly assigned to better organization and cooperation and partly to the psychological effect resulting from good ventilation. For instance, in the language of the average miner, he "tries to make a showing every day." In a comfortable atmosphere, he will take an interest in his work, and go about it in a more contented spirit than he could possibly do under hot, unfavorable, unhealthy, and depressing conditions.

A special effort, therefore, is constantly being made at the Copper Queen to increase the velocity of the air in all the raises, drifts, and stopes, and keep the ventilating system up-to-date with the newly developed workings.

CHANGES AFTER MECHANICAL VENTILATING SYSTEM WAS COMPLETED

Development Work and Ventilation System Planned Together

In general, the mine department makes an effort to plan development work in line with the ventilating system, as well as for prospecting purposes. The drifts are usually driven about 5 by 8 ft. and raises are run two compartments each, 50 in. in the clear. Recently standard raises have been adopted throughout the mine, and the timber compartments as well as manways have also been standardized. Rails are put over timber compartments to allow free passage of air (see Fig. 10) and manways have been made larger to permit more air to pass through them and at the same time to make it easier for the men to go up and down. Also when an orebody is found it is properly developed and prospected before stoping is commenced. In this way ventilation is really first established, while the orebody is being brought into a condition for stoping operations. This is a great help in the ventilation of stopes.

Square-Set and Shrinkage Stopes Ventilated Easiest

Recently other methods of stoping have been giving good results besides the original square set. Shrinkage, cut-and-fill, and top-slice stopes have progressed far enough to show that each method has its

particular application to certain kinds of ground. In ventilating stopes under these systems, probably the most easily ventilated are the square-set and shrinkage stopes. Next to these are the cut-and-fill, and the most difficult the top-slice stopes. However, in all cases at present where the top slice is being used, separate drifts and raises were driven to facilitate

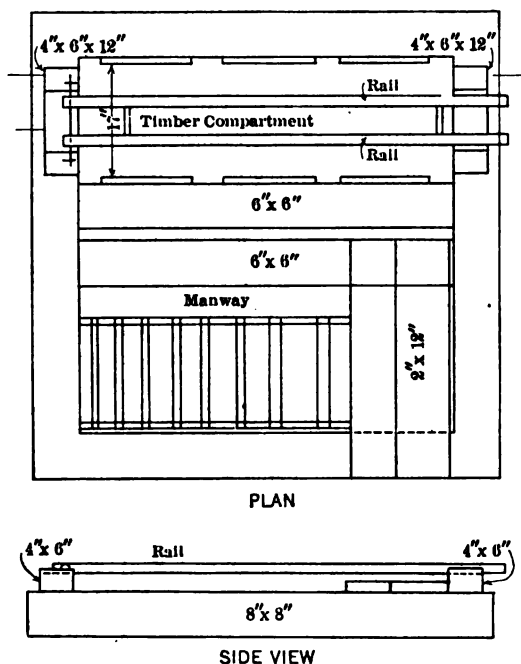


FIG. 10.—METHOD OF COVERING TIMBER COMPARTMENTS.

the handling of timber and tools and at the same time to give connections for ventilating purposes. Consequently, the top-slice stopes have about the same temperature and humidity as found in the square-set sections.

ADVANTAGES OF NEW VENTILATING SYSTEM

1. A smaller quantity of compressed air is used for ventilation.
2. All possible natural ventilation is utilized and the general direction of air currents is the same for both natural and mechanical ventilation.
3. There is a decrease in temperature and a lower percentage of humidity and CO_2 throughout the mine, which means healthier working conditions.
4. A saving of eight 5-h.p. blowers was made, as these have been removed from the mine.
5. The five lower divisions are under pressure, which partly acts as

a check in preventing some mine fires from crossing the Copper Queen side lines, as for example, the proximity of one mine fire near the Gardner workings.

6. Less decay of mine timbers.

7. The volume of air has been increased from 60 cu. ft. of air per man per minute, to about 300 cu. ft.

8. Formerly a large amount of moisture kept the timbers and working places in a damp condition and a species of fungi also covered most of the timbers. Since the mine fans were started, a large amount of water is removed daily from each division by the air currents so that the mine is comparatively dry and the fungi have disappeared.

9. The increase in velocity and volume of air resulted in the greater efficiency of the men.

10. The new ventilating system is advantageous to the mines adjacent to the Copper Queen mine; there is a great deal of broken and stoped ground along side lines permitting a large amount of air to escape through these avenues and ventilate stopes beyond the Copper Queen boundary.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should be in the form of a new paper.

The Formation of the Oxidized Ores of Zinc from the Sulphide

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(San Francisco Meeting, September, 1915)

CONTENTS

I. Introduction:

1. Subject and Scope.
2. Acknowledgments.
3. Review of Previous Work.
 - Field Data
 - Experimental Data
4. Plan of Present Investigation
 - Method of General Procedure

II. Experimental Work:

- A. The Solution of Sphalerite.
 1. The Solvent Effect of Iron Sulphates upon Sphalerite.
 2. The Solubility of Sphalerite in Salt Solutions of Silver, Copper and Lead.
- B. Formation of Zinc Carbonates.
 3. The Reactions of Carbonates and Bicarbonates With Solutions of Zinc Salts.
 4. The Reaction of Zinc Sulphate Solution With Limestones.
- C. Peculiar Conditions Affecting the Formation of Zinc Silicate (calamine) and Zinc Carbonate (smithsonite).
 - 5a. The Effect of Carbon Dioxide Upon the Basic Silicate and the Carbonates of Zinc in Water, and Its Bearing Upon the Conditions of Their Precipitation.
 - 5b. The Reactions of Solutions of Soluble Silicate With Calamine and Smithsonite in Solution in Presence of Carbon Dioxide.
 6. The Reactions of a Mixture of Hydroxide and Silicate of Sodium with a Dilute Solution of Zinc Sulphate.
- D. High-Temperature Methods.
 - 7a. The Influence of Temperature on the Formation of Calamine.
 - 7b. Possible Effect of Heat on Calamine and Smithsonite.

Summary of the Experimental Work.

III. Petrographic Work:

1. The Occurrence of Sphalerite in Limestones or in Cherts.
 - Summary of the Blende in Limestones and in Cherts.
2. The Partially Altered Sphalerite and the Oxidation Products.
 - a. Smithsonite with Blende.
 - b. Calamine with Blende.
 - c. Smithsonite and Calamine.
 - d. Summary on the Partially Altered Sphalerite and Its Oxidation Products.

IV. Conclusions.

- V. Photomicrographs and Photographs with Their Explanations.

I. INTRODUCTION

1. *Subject and Scope*

The formation of the oxidized ores of zinc has been recognized as a subject of great importance in economic geology. For many years these ores have been much sought, partly because they are more desirable for metallurgical treatment and partly because they are apt to be freer from gangue than is the sulphide of zinc. Up to very recent years the oxidized ores were the principal source of the metal and at present they still furnish a large portion of the commercial production.

The oxidized ores of zinc include six minerals, as shown in the following table:

Mineralogical Name	Chemical Name	Chemical Composition	Constituents, Per Cent.			
			Zn	SiO ₂	Fe	Mn
Calamine....	Basic silicate of zinc	2ZnO.SiO.H ₂ O or Zn-(OH) ₂ .ZnSiO ₃	54.20	25.0		
Smithsonite	Zinc carbonate.....	ZnCO ₃	52.00			
Hydrozincite	Hydro-carbonate of zinc.....	ZnCO ₃ .2Zn(OH) ₂				
Willemite...	Anhydrous silicate of zinc.....	2ZnO.SiO ₂ or ZnO.ZnSiO ₃	58.50	27.0		
Zincite.....	Zinc oxide.....	ZnO.....	80.30			
Franklinite..	Iron-manganese-zinc oxide.....	(FeZnMn)O(FeMn) ₂ O ₃	12.00		42.0	15.0

Only the first three, calamine, smithsonite and hydrozincite have been considered in this investigation. Wherever the terms "oxidized ores" and "oxidation products" are used in this paper they are, unless otherwise indicated, strictly limited to these three ore-minerals of zinc.

These oxidation products are universally believed to be derived from the sulphide of zinc, as their original mineral. The chemical reactions of the change, however, are not fully understood and have not, heretofore, been studied experimentally in the light of physico-chemical principles. The influence of oxidation by surface waters and of other metallic compounds which are associated with the blende, is only inferred from field relations. At the suggestion of Dr. J. F. Kemp, the writer undertook a series of experiments with the hope of clearing up some important points in connection with the chemistry of the precipitation of the three oxidized ores and in particular, of establishing the peculiar set of conditions under which the antecedent oxidation of the blende has taken place.

The principal part of the experiments is, in accordance with the geological evidence, limited to wet methods, and deals with a single substance, or a group of substances in solution, which are capable of effect-

ing the solution of the blende or of converting its oxidation product into a carbonate or a basic silicate, as the case may be.

2. Acknowledgments

The writer takes great pleasure in acknowledging his indebtedness to his professors and friends for aid received in the preparation of this paper. First among them is Professor J. F. Kemp, of Columbia University, under whom this work was done. Throughout the entire laboratory work, and especially on the experimental side, he has manifested great interest in the results as they appeared. Whatever merits this paper may have are largely due to his valuable assistance and advice. For many useful criticisms relative to the petrographic feature of the work, the writer is indebted to Professor C. P. Berkey, of the same University. The writer's thanks are also due to Professor T. C. Chamberlin, of the University of Chicago, for inspiration and encouragement through correspondence and otherwise during the execution of this research, and to Dr. R. T. Chamberlin for many helpful suggestions. To Professor R. J. Colony of Cooper Union of New York City, the writer is indebted for checking the results of optical examinations of the crushed fragments. Acknowledgments to other friends have been indicated in the course of the following pages.

3. Review of Previous Work

Field Data.—Field data concerning the paragenesis of the oxidized ores of zinc are abundant. The most suggestive contribution to the literature of the subject is by Dr. R. Beck.¹ In the discussion of the zinc deposits at Raibl in Carinthia, Austria, he regards the oxidized ores of zinc in that region as the results of extensive metasomatism, and says that the limestone, the inclosing rock of the ores, "is directly replaced by calamine and the structure of the original rock is retained by the calamine and carbonate of zinc." Our general experience shows that calamine and smithsonite are derived from blende. Therefore, the solutions which replaced the soluble limestone presumably obtained their salts by the oxidation of sphalerite and the process was essentially one of secondary enrichment. That is to say, the oxidation and solution of the sphalerite must equally depend: (1) on the action of the oxidation products of pyrite or marcasite on the sphalerite in presence of water (these iron sulphides in nature are invariably associated with zinc sulphide); (2) on the direct oxidation of sphalerite as a result of electrolytic action²

¹ R. Beck. *Lehre von den Erzlagerstätten*, Band 2, p. 596 (1901).

² V. H. Gottschalk, and H. A. Buehler. Oxidation of Sulphides, *Economic Geology*, vol. vii, p. 28 (1912).

TABLE I.—*Analyses of Mine Waters and Well Waters in Parts per Million*

	1	2	3	4	5	6	7	8	9	10
S*.....	184.2	1886.0	2033.4	3200.3	—	221.4	429.0	154.3	29.7	161.0
Cl.....	0.0	0.0	0.0	0.0	0.0	0.0	12.8	67.45	3.5	—
Free										
H ₂ SO ₄ ...	0.0	220.5	39.0	1568.0	333.2	0.0	0.0	0.0	0.0	0.0
Combined										
H ₂ SO ₄ ...	0.0	0.0	6060.9	8032.9	0.0	0.0	0.0	0.0	0.0	0.0
Metallic										
Fe.....	0.0	670.0	362.0	523.5	465.0	—	0.0	—	1.0	15.5
Fe.....	0.0	490.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	—
Fe.....	0.0	180.0	293.0	1091.5	—	0.0	0.0	3.0	—	—
CaO.....	354.1	—	700.5	—	—	—	501.0	333.9	—	—
MgO.....	31.4	—	0.0	—	—	—	—?	108.6	—	—
Pb.....	0.0	0.0	0.30	0.0	0.0	0.0	0.0	0.0	1.3	0.0
Metallic										
Zn.....	20.0	1432.5	80.50	2250.0	910.0	20.9	257.0	1.7	7.3	70.0
Acidity						Alkaline				

	11	12	13		14	15	16	17
KCl.....	9.5	6.1	0.0	SO ₄	233.4	222.4	215.9	104.0
K ₂ SO ₄	0.0	0.0	0.0	SiO ₂	6.8	8.0	2.1	18.0
NaCl.....	0.0	1.3	4.5	Al ₂ O ₃	2.8	0.7	2.8	0.0
Na ₂ SO ₄	130.3	38.5	30.8	FeO&Fe ₂ O ₃	6.7	9.0	6.8	9.5
Al ₂ (SO ₄) ₃ ...	68.3	73.6	3.3	MnO	2.7	5.9	8.3	0.0
CaSO ₄	1528.0	924.6	371.5	ZnO	23.5	5.9	3.5	11.1
Ca(HCO ₃) ₂	0.0	0.0	0.0	Cl	trace	trace	trace	12.4
MgSO ₄	trace	192.6	60.4	CaO	148.5	101.4	127.0	65.0
FeSO ₄	611.9	387.7	2.6	MgO	28.4	24.7	21.7	24.9
PbSO ₄	0.04	0.0		Insoluble				
ZnSO ₄	586.2	851.7		residue....	17.5	15.8	21.0	0.0
SiO ₂	23.2	23.2	7.4					
H ₂ SO ₄	170.9	251.7	0.0					
CO ₂ (free)...	12.0	87.0	19.7					

1. Oronogo Circle mine, Oronogo, Mo.

2. Duenweg mine, Joplin district, Mo.

3. Continental mine, Joplin district, Mo.

4. Mary mine, Joplin district, Mo.

5. Diamond Jack mine, Cartersville, Mo.

6. 9th Shaft, Oronogo Circle mine, Oronogo, Mo.

7. Well on Centre Creek.

8. Well on Main Street, Joplin, Mo.

9. Centre Creek water.

10. Centre Creek water

11. Arkansas mine, Joplin district, Mo.

12. Victor mine, Joplin district, Mo.

13. O. K. mine, Joplin district, Mo.

14–17. Mine waters of Rothschoenberger Stollen, near Freiberg, Germany.

— = No determination was made of the element.

O = The element was absent.

* The sulphur in all cases exists as sulphate.

due to the contact of sphalerite and iron sulphides; or (3) on both of these actions.

In addition to this statement of Beck's, we have the evidence afforded by analyses of mine waters and of well waters from zinc districts.

"While the waters taken from a mine may differ slightly in composition from the ground waters that seeped through the undisturbed deposit, in that they are probably more dilute on account of the freer circulation of the solutions, and are also more highly oxidized on account of greater access to the atmosphere, yet nevertheless their composition approaches fairly closely that of the original ground-waters."³

The Table I comprises typical analyses⁴ of the waters from zinc mines both in Europe and in America, and of the waters from wells in the zinc district of Joplin, Mo. The figures given are all in parts per million.

On inspection of these data, it will be noted that sulphur in all cases exists as sulphate, indicating as a distinctive feature of these waters, the predominance of the sulphates of zinc, iron, calcium and generally magnesium, over other salts. It has also been noted, according to Mr. Waring,⁵ the Chemist of Webb City, Mo., that the iron in the waters freshly drawn from the depths of the Joplin mines, is always present as ferrous sulphate, while the waters from the surface levels frequently contain ferric sulphate. Further evidence has recently been shown by Hodge⁶ as to the presence of ferrous salts in the waters at some depth and of ferric near the surface.

According to Allen⁷ and Wells,⁸ the excess of sulphuric acid formed by the oxidation of the iron sulphides is responsible for the increased solubility of the metallic sulphides in association. This assumption does not seem to be entirely consistent with the experimental facts,⁹ or to be generally supported by chemical analysis. Mr. Waring,¹⁰ speaking from his wide experience in the zinc mines of Joplin district, Mo., says that free sulphuric is not always present; in some cases, the waters that are quite rich in zinc sulphate contain no sulphuric acid. On the contrary they gave a strong alkaline reaction to methyl orange, frequently containing considerable amount of calcium bicarbonate. The presence of the latter compound in solution may be a factor in the process of the

³ N. C. Cooke. *Journal of Geology*, vol. xxi, p. 5 (1913).

⁴ For the analyses Nos. 1 to 10, the writer is indebted to W. G. Waring of Webb City, Mo., who has very kindly placed at his disposal the data which for the first time appears in print. For Nos. 11 to 13 inclusive, the writer has quoted from a paper by Messers Buehler and Gottschalk, *Economic Geology*, vol. v, p. 34 (1910); Nos. 14 to 17 from *Lehre von den Erzlagerstätten*, R. Beck, vol. ii, p. 402 (1901).

⁵ Private communication.

⁶ E. T. Hodge. *Economic Geology*, vol. x, p. 123 (1915).

⁷ E. T. Allen. *Economic Geology*, vol. v, p. 387 (1910).

⁸ R. C. Wells. *Economic Geology*, vol. v, p. 480 (1910).

⁹ Gottschalk and Buehler. *Economic Geology*, vol. vii, p. 27 (1912).

¹⁰ Private communication.

formation of zinc carbonate and will be discussed experimentally later in this paper.

Experimental Data.—In the literature of this subject no experimental data bearing on this problem were found. Those who have dealt with the question in the laboratory have rarely worked under the conditions which commonly obtain in nature; *i.e.*, low temperatures, low pressures, small concentrations, and the combination of all these conditions, presenting a narrow range of possibilities for imitative synthetic studies. H. de Senarmont¹¹ showed that zinc carbonate can be formed from zinc chloride and calcium carbonate at 150°C. Gorgeu¹² in 1887 claimed that he obtained the anhydrous silicate and the oxide of zinc (willemite and zincite) by fusing a mixture of chemical compounds containing the required constituents. His experiments have been repeated by the writer, but without producing the same results.¹³ No attempt has heretofore been made, even using high temperature, to produce synthetically the basic silicate of zinc (calamine).

The investigations of Allen and Creushaw¹⁴ on zinc compounds have no bearing on this problem, since their work was confined to the study of zinc sulphides.

4. Plan of Present Investigation

The present work was divided into two parts, experimental work and microscopic studies of thin sections. Experimentally the main purpose was to find: (a) what is the exact nature of the compound or compounds, commonly associated with the sulphide of zinc, which cause it to be oxidized and carried into solution in meteoric waters; (b) what are the most common reagents or compounds which are likely to have caused the precipitation of carbonates or the basic silicate of zinc; (c) what are the peculiar conditions under which these oxidation products may be obtained in the laboratory; (d) to what extent the formation of these oxidation products is limited by ordinary laboratory conditions; and (e) what influence is exerted by temperature. The microscopic study was done chiefly in the hope of obtaining further evidence throwing more light on the problem.

The experimental work here described deals with the following questions:

1. The solvent effect of iron sulphates on sphalerite.
2. The solubility of sphalerite in a salt solution of silver, of copper and of lead.

¹¹ Doelter y Asterich. *Handbuch der Mineralchemie*, Band 1, p. 499.

¹² Bulletin 10, *Société Française de Mineralogie*, pp. 30 to 39.

¹³ See the discussion under "The Influence of Temperature on the Formation of Calamine," Experiment 12.

¹⁴ *American Journal of Science* 4 series, vol. xxxiv, pp. 341 to 360 (1912).

3. The reactions of carbonates and bicarbonates with solutions of zinc salts.

4. The reaction of zinc sulphate solution with pure and impure limestones.

5. The effect of carbon dioxide upon the basic silicate and the carbonates of zinc in water, and its bearing upon the conditions of their precipitation.

6. The reactions of a mixture of hydroxide and silicate of sodium with dilute solutions of a zinc salt.

7. The influence of temperature on the formation of calamine, and its possible effects on natural calamine and smithsonite.

On account of the wide range of the subjects involved in the experiments, each series will be described and discussed under its own heading. Unless otherwise indicated, all of these experiments were conducted under ordinary conditions of temperature and pressure. Concentrations of solutions were small, deci-normal in some cases, and fiftieth in others; these concentrations may slightly differ in behavior from those commonly obtained from the ground waters, but the difference is only in the speed, and not in the nature, of the chemical reactions.

The paper also includes a discussion, based on the suggestion of Dr. Kemp¹⁶ as to the origin of the zinc deposits at Franklin Furnace, and Ogdensburg, N. J., of the effect of heat on calamine and smithsonite, with possible changes to willemite and zincite respectively.

Method of General Procedure.—Unless otherwise indicated, the mineral to be tested was first powdered; and only the portion which passed 50- or 100-mesh sieve, as the case might be, was used. Chemically made zinc compounds were also used and are indicated in each case. The amount of reagents in solution, as to their constituents, were definitely known whenever it was deemed necessary. In all cases, the precipitate, the residue if any, and the liquid contents in filtrates, were determined quantitatively. The comments which usually follow the record of the experiment were made in each case in consequence of the laboratory results. In determining the temperatures of formation of the oxidation products (here including willemite and zincite but excluding hydrozincite) both the quantitative analysis and microscopic examination of the crushed fragments were employed.

II. EXPERIMENTAL WORK

A. *The Solution of Sphalerite*

1. *The Solvent Effect of Iron Sulphates upon Sphalerite.*—As already remarked, there is commonly present in the freshly drawn water from the depths of zinc mines, such as those of Joplin, an iron salt in solution,

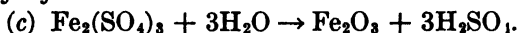
¹⁶ *Transactions of New York Academy of Science*, vol. xiii, p. 92.

which, if drawn from some depth, is invariably in the form of ferrous sulphate, and doubtless indicates the oxidation of the blende to some extent at the expense of the reduction of ferric sulphate.

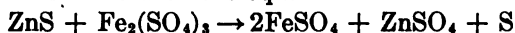
The ferric sulphate is presumably produced by the oxidation of an iron bisulphide (pyrite or marcasite) which occurs not infrequently in the zinc-producing districts. The reaction is ordinarily written:



(b) $2\text{FeSO}_4 + \text{H}_2\text{SO}_4 + \text{O} \rightarrow \text{Fe}_2(\text{SO}_4)_3 + \text{H}_2\text{O}$. This ferric solution may hydrolyze to



Reaction (c), however, occurs only in the absence of such minerals as blende. In blende-bearing strata the reaction (c) does not occur; at least, no such case has ever been reported. On the contrary, as shown by experimental results recorded below, when this strongly oxidizing agent is held in waters which percolate into blende-bearing rocks, it will react with the sphalerite and allow zinc to pass into solution. The reaction is in accordance with the equation:



In order to determine just what proportion of the ferric sulphate is necessary to oxidize one part of sphalerite, the following experiments were undertaken:

Experiment 1.—A piece of sphalerite obtained from Joplin, Mo.,

	I (a)		I (b)		II	III		IV	
	Actual amounts originally present in grams				Conditions in the filtrate before analysis. Grams.	Filtrate		Residue	
	In solution		In solid						
	Element	Compound	Element	Compound		Grams	Per Cent.	Grams	Per Cent.
S.....			1.598					0.3055	18.84*
Impurity.....								0.0132	
Zn.....			3.262					trace	
Fe ⁺⁺						0.8794	6.304		
Fe ⁺⁺⁺	1.8636					0.9833	7.049		
SO ₄	4.8027					8.8270	63.270		
Zn.....						3.2550	23.330		
FeSO ₄					2.3900				
Fe ₂ (SO ₄) ₃		6.6663			3.5190				
ZnSO ₄					8.0350				
ZnS.....				4.860					
Total...	6.6663	6.6663	4.860	4.860	13.9447	13.9447	99.953	0.3187	18.84

*This shows that the rest of the sulphur (81.16 per cent.) has been oxidized into sulphate.

and known to contain 0.9972 g. ZnS for every gram of the sample was ground to pass a 50-mesh sieve; 4.873 g. of this material (on account of its impurity, this amount is a little less than $\frac{N}{10}$ of its molecular weight) was digested in 1,000 c.c. ferric sulphate solution, containing 6.663 g. of the salt (approximately $\frac{N}{10}$). The reaction is manifested as soon as the sphalerite is placed in the ferric solution. The mixture was allowed to stand in a 2-liter bottle, whose mouth was closed with a bunsen valve, and also attached by a side glass tube to an air tank from which ordinary air was allowed to pass slowly into the mixture 15 min. at a time, three times a day, for a period of four weeks. The foregoing table shows the analyses of both the residue, which weighed 0.3204 g. and the filtrate.

Experiment 2.—An equal amount, 4.873 g. of lump sphalerite from the same sample was treated in 1,000 c.c. $\frac{N}{10}$ ferric sulphate solution. In the same way as in Experiment 1, ordinary air was allowed to pass slowly into the mixture three times a day, 15 min. at a time, for a period of four weeks. The residue weighed 2.0154 g. The following analyses were obtained from the residue and the filtrate:

	I (a)		I (b)		II	III		IV	
	Actual amounts originally present in grams				Conditions in the filtrate before analysis. Grams	Filtrate		Residue	
	In solution		In solid						
	Element	Compound	Element	Compound		Grams	Per Cent.	Grams	Per Cent.
S.....			1.598				0.8572	53.23*	
Impurity.....							0.0136		
Zn.....			3.262				1.1490		
Fe ⁺⁺									
Fe ⁺⁺⁺	1.8636					1.1210	10.28		
SO ₄	4.8027					0.7413	6.795		
Zn.....						6.9380	63.59		
FeSO ₄					3.047	2.1120	19.36		
Fe2(SO ₄) ₃		6.6663			2.6523				
ZnSO ₄					5.213				
ZnS.....				4.860					
Total...	6.6663	6.6663	4.860	4.860	10.9123	10.9123	100.025	2.0198 53.23	

* This shows that the rest of the sulphur (46.77 per cent.) has been oxidized into sulphate.

In both filtrates the iron and sulphate contents were determined with a known strength of K₂Cr₂O₇ and BaCl₂ solutions respectively. The

amount of the sulphates of ferrous and ferric, together with that of zinc, in the filtrate was recalculated and recorded in column II. These analyses also show an excess of sulphur in the residue, indicating the presence of sulphuric acid, which is sometimes present with FeSO_4 and ZnSO_4 in the waters of zinc mines.¹⁶

According to the above results, there was practically no zinc remaining in the residue in experiment 1, and the amount of ferric (47.21 per cent.) reduced to ferrous, as a result of the oxidation of ZnS , in the first case is 12.96 per cent. less than in experiment 2, in which, in spite of the presence of more ferrous (60.17 per cent.) than ferric, only 65.34 per cent. of ZnS was oxidized. This difference is believed, other things being equal, to have been affected by the fine grinding of the ZnS in the first case, and its absence in the other.¹⁷

Whatever values are given ultimately for the rate of oxidation of zinc sulphide in the presence of ferric sulphate, the results of the above experiments show beyond any doubt that the chemical equation $\text{Fe}_2(\text{SO}_4)_3 + \text{ZnS} \rightarrow 2\text{FeSO}_4 + \text{ZnSO}_4 + \text{S}$ merely represents the chemical reaction and not the ratio. It is further shown that in contact with the oxygenated water of meteoric origin, which is no doubt an important factor in nature, even a comparatively small proportion of ferric sulphate will accelerate the rate of oxidation of the sulphide of zinc very materially. These apparent inconsistencies with the demands of the solution theory as required by the above chemical equation may be attributed to the influence of electrolytic action, as in the case with two different sulphides.¹⁸

The effect of ferrous sulphate upon ZnS is quite different from that of ferric sulphate. A solution of ferrous sulphate in contact with sphalerite produced no change whatsoever in either, if air is absolutely excluded. Different effects are observed, however, when the mixture is freely exposed to air. The following experiments were undertaken to demonstrate these points:

Experiment 3.—In this experiment the ferrous sulphate solution was obtained by allowing pyrites to oxidize under the influence of water and air and leaching the residue. The liquor was treated with scrap iron, and the neutral solution evaporated until a hydrate ($\text{FeSO}_4 \cdot 7\text{H}_2\text{O}$) is deposited. The salt is green in color. Two portions of it, containing approximately 3.799 g. FeSO_4 , without water, were treated in 500 c.c. of distilled water in two separate bottles (a) and (b), and each solution had a strength about $\frac{N}{10}$ of its molecular weight. The solution was then mixed with 4.873 $\frac{N}{10}$ g. of sphalerite, previously ground, and treated with 500 c.c. water.

¹⁶ See analyses 2, 3, 4, 5, in Table I.

¹⁷ *Economic Geology*, vol. v, p. 35 (1910).

¹⁸ *Economic Geology*, vol. vii, p. 28 (1912).

The mouth of the bottle (a), after mixing, was closed with its glass stopper, and the solution was left in the bottle undisturbed for four weeks. In (b) the water used was collected from rain and the mixture was allowed to stand in the bottle uncovered and freely exposed to air for the same period of time.

At the end of this period, each mixture was filtered separately. Both the residue and the filtrate in each case, were subjected to analysis.

In the analysis of (a) the filtrate was analyzed for Fe and Zn, but no Zn was found and only ferrous salts were present in solution indicating that no oxidation had occurred. The residue was still in the form of ZnS, as revealed by qualitative tests.

The analysis of (b) gave, from the residue,

	Grams	Grams	Grams
Fe ₂ O ₃	0.7343 = Fe	0.2569 = Limonite	= 0.9824
S	1.3671 = S	1.3671 = Sulphur	= 0.1554
Zn	2.4710 = Zn	2.4710 = ZnS	= 3.6827
Impurities	0.0137 = Imp.	0.0137 = Imp.	= 0.0137
H ₂ O	0.2251 = H ₂ O	0.2481*	
	4.8112		4.8342

and from the filtrate,

Fe	1.1410	= FeSO ₄	= 3.1030
SO ₄	3.1142		
Zn	0.7846	= ZnSO ₄	= 1.4368
	5.0398		5.0398

The Fe₂O₃ is a yellowish red precipitate in the flask. After filtering, it was added to the residue. It is believed to be a hydrated iron oxide such as limonite, since the amount of water found in the residue is only 10 per cent. less than what is theoretically required for the amount of Fe₂O₃ (0.7343) to be limonite. The results given above, show: (1) the partial oxidation of sphalerite due to free access of air; (2) that the ferrous sulphate may, under the influence of air, be oxidized to hydrated ferric oxide, and enable more sphalerite to be oxidized by giving up its sulph-oxygen ions to the latter, as evidenced by the free sulphur recalculated in the mineral component column for the residue; (3) that ferrous sulphate does not oxidize sphalerite—the analysis of (a)—but if it has free access to air, it may form hydrated ferric oxide and subsequently oxidize all the Zn into sulphate, leaving its sulphur in the free state, if the mixture has been allowed to stand long enough in the air; (4) that the oxidation of ferrous to ferric by air in contact with sphalerite is probably exhibited, as in the case of pyrites with sphalerite;¹⁹ and (5) that, with

* This figure is calculated from the formula of limonite, assuming that the Fe is in the form of that mineral.

¹⁹ V. H. Gottschalk and H. A. Buehler. *Economic Geology*, vol. vii, pp. 22, 28 (1912).

sufficient time, all ZnS will go into solution as sulphate, and all the ferrous iron will oxidize to ferric oxide, which in the presence of water yields limonite or some other iron hydrate.

2. *The Solubility of Sphalerite in Salt Solutions of Silver, of Copper and of Lead.*—That zinc can be replaced or carried from its mineral in form of sulphide into solution to form a salt by the replacing action of a number of metallic elements in the form of their respective salts in solution has long been known to physical chemists. In 1888, Schuermann²⁰ established a series in which he showed that on application of heat the zinc as sulphide is dissolved in a solution of a soluble salt of any metal higher in the series, and that zinc can be precipitated as sulphide from a solution of its salt at the expense of a sulphide of any metal lower in the series. The solutions used in his experiments were all sulphates, nitrates, or chlorides. The following series was established in the order of their desire to combine with sulphur:

1, Pt; 2, Hg; 3, Ag; 4, Cu; 5, Bi; 6, Cd; 7, Sb; 8, Sn; 9, Pb; 10, Zn; 11, Ni; 12, Co; 13, Fe; 14, As; 15, Tl; 16, Mn. The first nine metals may be considered as dissolving agents and the last six as precipitants of the zinc sulphide. The reaction may be expressed in one case, thus: $\text{MSO}_4 + \text{ZnS} \rightarrow \text{ZnSO}_4 + \text{MS}$; and in the other case thus: $\text{MS} + \text{ZnSO}_4 \rightarrow \text{ZnS} + \text{MSO}_4$; M representing any metal which either precedes or follows zinc in the series.

The metals whose minerals are commonly found in geological association with those of zinc, are iron, copper, cadmium, lead, and occasionally silver. All these metals except the first precede zinc in Schuermann's series. Electrolytically, however, all of them can be displaced by zinc, since the electromotive force of the current required to deposit each metal is less than that for the metal preceding in the electromotive series.²¹

Some results of scientific interest might be obtained by treating the sulphide of zinc electrolytically in a salt solution of any metal preceding zinc in Schuermann's series; but this is outside our present purpose. It is not without interest, however, to consider the other possible reasons for the solubility of zinc sulphide in the solution of a salt of any one metal above zinc in this series, without attributing the reaction entirely to their affinity for sulphur, as did Schuermann. With this in mind, a series of experiments were undertaken at different temperatures, degrees of concentration and time limits, in order to ascertain their relative importance, each to the other, and to observe in each case what change would take place under ordinary pressures. For this purpose two sulphates and one nitrate were taken—the three metals of which, silver, copper and lead, having, according to Schuermann, great affinity for sulphur.

²⁰ Ernest Schuermann. Über die Verwandtschaft der Schwermetalle zum Schwefel. *Liebig's Annalen der Chemie*, vol. ccxlvii, p. 326 (1888).

²¹ Alex. Smith. *General Inorganic Chemistry*, p. 362.

Throughout the series of experiments a mixture of definite quantity in equal or less molecular ratio of each solution was used. In each case the mixture was subject to various temperatures for a given time, and to a definite temperature for different periods of time, until the desired reaction was obtained. The solution of each metallic salt was kept at a uniform temperature before the beginning of each experiment, and the sphalerite used in all these experiments, unless indicated otherwise, was ground until it passed a 50-mesh sieve.

Experiment 4.—This comprised three tests, as follows:

(a) ZINC SULPHIDE WITH A SOLUTION OF SILVER SULPHATE

	Molecular Concentration				Temp., °C.	Time	Reaction
	ZnS		Ag ₂ SO ₄				
	Grams	E. N.	C. C.	Norm.			
1	0.4873	N 10	100	N 25	15	4 hr.	None
2	0.2436	N 20	100	N 25	25	4 hr.	Slight
3	0.1950	N 25	100	N 30	30	4 hr.	Slight
4	0.1620	N 30	100	N 35	45	5 min.	Complete
5	0.1220	N 40	100	N 40	45	5 min.	Complete
6	0.1220	N 40	100	N 50	50	4 min.	Complete
7	0.4873	N 10	100	N 25	40	6 min.	Complete
8	0.4873	N 10	100	N 25	35	25 min.	Apparent, but not complete
9	0.4873	N 10	100	N 25	30	1 hr.	Slight
10	0.4873	N 10	100	N 25	25	3 hr.	Slight
11	0.4873	N 10	100	N 25	20	4 hr.	Slight
12	0.4873	N 10	100	N 25	0	20 hr.	None
13*	4.873	N 10	1,000	N 25	20	20 hr.	None
14*	4.873	N 10	1,000	N 25	30	12 hr.	Slight

E. N. = Equivalent normality. Norm. = Normality. Natural zinc sulphide and chemically pure silver sulphide were used.

* The sphalerite in this test was in its lump form without grinding.

(b) ZINC SULPHATE WITH A SOLUTION OF COPPER SULPHATE
($\text{CaSO}_4 \cdot 5\text{H}_2\text{O}$)

	Molecular Concentration				Temp. °C.	Time	Reaction
	ZnS		CuSO ₄				
	Grams	E. N.	C. C.	Norm.			
1	0.4873	N 10	100	N 10	15	4 hr.	None
2	0.2436	N 20	100	N 20	25	4 hr.	None
3	0.1950	N 25	100	N 25	30	4 hr.	Slight
4	0.1620	N 30	100	N 30	45	7 min.	Almost complete
5	0.1220	N 40	100	N 40	45	7 min.	Almost complete
6	0.1220	N 40	100	N 40	50	6 min.	Complete
7	0.4873	N 10	100	N 10	40	9 min.	Complete
8	0.4873	N 10	100	N 10	35	25 min.	Slight
9	0.4873	N 10	100	N 10	30	2 hr.	Slight
10	0.4873	N 10	100	N 10	25	5 hr.	Slight
11	0.4873	N 10	100	N 10	20	13 hr.	Slight
12	0.4873	N 10	100	N 10	0	13 hr.	None
13*	4.873	N 10	1,000	N 10	20	13 hr.	None
14*	4.873	N 10	1,000	N 10	30	15 hr.	Slight

E. N. = Equivalent normality. Norm. = Normality. Natural zinc sulphide and chemically pure copper sulphide were used.

* The sphalerite in this test was in its lump form without grinding.

(c) ZINC SULPHIDE WITH A SOLUTION OF LEAD NITRATE**

Molecular Concentration					Temp., °C.	Time	Reaction
ZnS		Pb(NO ₃) ₂					
Grams	E. N.	C. C.	Norm.				
1	0.4873	N 10	100	N 10	15	4 hr.	None
2	0.2436	N 20	100	N 20	25	4 hr.	None
3	0.1950	N 25	100	N 25	30	4 hr.	Very slight
4	0.1620	N 30	100	N 30	45	13 min.	Complete
5	0.1220	N 40	100	N 40	45	13 min.	Complete
6	0.1220	N 40	100	N 40	50	11 min.	Complete
7	0.4873	N 10	100	N 10	40	18 min.	Complete
8	0.4873	N 10	100	N 10	35	30 min.	Slight
9	0.4873	N 10	100	N 10	30	3 hr.	Slight
10	0.4873	N 10	100	N 10	25	7 hr.	Slight
11	0.4873	N 10	100	N 10	20	14 hr.	Slight
12	0.4873	N 10	100	N 10	0	14 hr.	None
13*	4.8730	N 10	1,000	N 10	20	14 hr.	None
14*	4.8730	N 10	1,000	N 10	30	20 hr.	Very slight

E. N. = Equivalent normality. Norm. = Normality. Natural zinc sulphide and chemically pure lead nitrate were used.

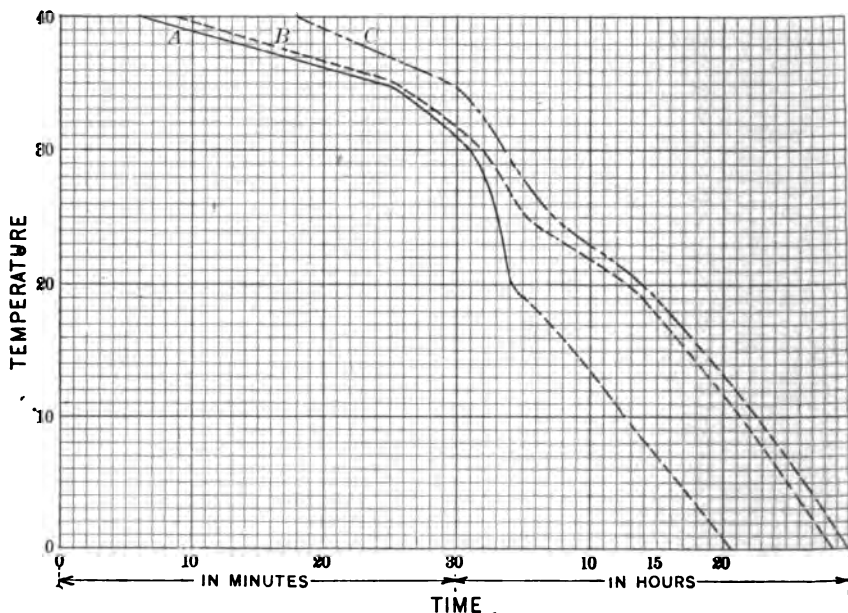
In explanation of the results in the above three tables, it should be mentioned that the precipitated sulphides of Ag, Cu and Pb were all amorphous and that all were qualitatively determined. No attempt was made for quantitative determination.

From these results it can be readily seen that temperature alone has played a most important rôle in the solubility of ZnS at its various mol-concentrations. This is best brought out by numbers 6 to 14 inclusive

* The sphalerite in this is in its lump form without grinding.

** It was also tried with PbSO₄, but this compound is exceedingly difficult to dissolve, so Pb(NO₃)₂ was employed instead.

in each case. It is also interesting to note from a scientific point of view, that the temperature of 45°C. seems to be a critical one in all cases. The differences in the amount of ZnS and in the concentration of the reagent do not exert any appreciable influence upon the time required to complete the reaction. The differences in time required for the solution of ZnS in each of these different solutions at different concentrations is apparently due to the difference of their metallic sulphides either in potential values or in solubility products, or both. The single potentials



From 0 to 30 every small square represents one min.; from 30 to 20 to the right every small square represents one hr.

Curve A represents the reaction between ZnS and Ag₂SO₄.

Curve B represents the reaction between ZnS and CuSO₄.

Curve C represents the reaction between ZnS and Pb(NO₃)₂.

The broken part in each curve represents the unknown time required at a temperature between 20° and 0°C. Their concentrations are indicated in their respective tables.

FIG. 1.

of the metals, Ag, Cu and Pb, in solution saturated with their sulphides, as determined by Zengelis²² are as follows:

Ag in Ag₂S - 0.34 volts

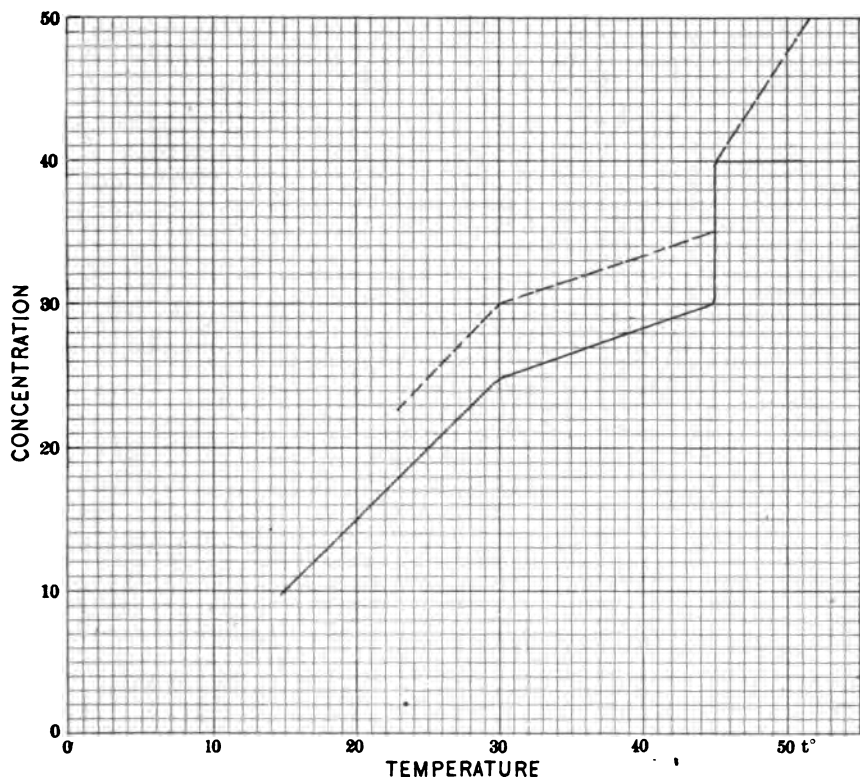
Cu in Cu₂S - 0.203 volts

Pb in PbS - 0.20 to 0.06 volts

The writer is unable to find any information concerning the exact potential value of zinc under similar conditions; but it is readily inferred

²² K. Zengelis. *Ueber die electromotische Kräfte unlöslicher und komplexer Salze.*

that the potential of zinc in its zinc sulphide would be lower in comparison with that of any of these three metals from the fact that their sulphides can be precipitated from their salt solutions in contact with zinc sulphide at a definite temperature, as shown by the above experimental results. Their solubility product doubtless plays an equally important part in these reactions. The fact that less soluble compounds must be thrown down in presence of more soluble ones is too well known to be doubted.



The dotted line represents the behavior of Ag_2SO_4 solution with ZnS . The continuous black line represents the general behavior of CuSO_4 and $\text{Pb}(\text{NO}_3)_2$ solutions with ZnS .

Note the influence of these three solutions on ZnS at the temperature of 45° irrespective of their concentrations.

FIG. 2.

Fig. 1 is intended to show the time required for the reaction at various temperatures between each of these three solutions and ZnS .

Fig. 2 illustrates the general behavior of ZnS with two different salt solutions of these three metals at different concentrations and temperature, irrespective of the time required for the reaction in each case.

The above results may help to account for the fact that when zinc occurs in paying quantities as sulphide with the sulphide of Ag , Cu , or

Pb, the zinc always appears most abundantly only beneath the horizon of the minerals of the latter three, if they occur contemporaneously in the same horizon. It must not be understood that sphalerite does not occur side by side with galena or chalcopyrite or argentiferous galena. When they do occur together, it simply means there was enough sulphur to take care of all of them. This relation is particularly well shown in the occurrence of zinc-lead deposits in the Mississippi Valley, where, if galena is the chief ore, ZnS occurs only beneath the richest horizon of galena; if ZnS is the principal ore, PbS is found only in the upper portion of the orebody, as is the case with the ores in Joplin, Mo. The zinc mines at Butte²² at the present time also show this to be true. In former times, silver sulphide occurred and was mined in the upper horizon. At present, blende is mined beneath that horizon. It is not unlikely that in these mines at Butte, copper sulphide may also occur at a greater depth in the future, but this would simply mean that the occurrences represent different periods of deposition.

With these geological observations on the one hand, and chemical relations on the other, there can be no doubt that ZnS , if present in solution together with sulphides of Pb, Cu, or Ag, is the last mineral to be formed in nature.

B. Formation of Zinc Carbonates

3. *The Reactions of Carbonates and Bicarbonates With Solutions of Zinc Salts.*—From the fact that the molar solubility of ZnCO_3 is much lower than that of ZnSO_4 , or ZnCl_2 , and that ZnS is often replaced in nature by ZnCO_3 , it is readily inferred that the substances in an orebody, capable of changing ZnS to ZnCO_3 , or precipitating ZnCO_3 from a solution in which Zn is held as a sulphate or chloride, are largely the soluble normal carbonates and bicarbonates.

The normal carbonates of sodium and potassium and bicarbonates of sodium and calcium were used for the following experiments to demonstrate this point.

Experiment 5.—On the precipitation of ZnCO_3 from its soluble compounds.

(a) Normal carbonate of sodium: 400 c.c. $\frac{N}{50}$ solution of ZnSO_4 was mixed with an excess of Na_2CO_3 solution of same normality. A flocculent precipitate formed immediately. Similarly 400 c.c. $\frac{N}{50}$ solution of ZnCl_2 was mixed with an excess of Na_2CO_3 solution and a precipitate likewise formed immediately. Each of these mixtures ($\text{ZnSO}_4 + \text{Na}_2\text{CO}_3$ and $\text{ZnCl}_2 + \text{Na}_2\text{CO}_3$) was then warmed and each precipitate was filtered and washed until the washings did not show any presence of Cl or SO_4 .

²² Learned orally from Prof. J. F. Kemp, who has just visited some of the zinc mines at Butte, Mont.

The precipitate from $(\text{ZnSO}_4 + \text{Na}_2\text{CO}_3)$ weighed 0.4423 g. and that from $(\text{ZnCl}_2 + \text{Na}_2\text{CO}_3)$ 0.4322 g. Both were amorphous and each yielded the following after being dried in the oven and cooled in the desiccator:

The precipitate from the reaction $\text{ZnSO}_4 + \text{Na}_2\text{CO}_3$:

Chemical Components	Grams	Per Cent.	Molecular Weight	Molecular Ratio
ZnO.....	0.33320	75.34	81.40	0.9255
CO ₂	0.06003	13.57	44.00	0.3084
H ₂ O.....	0.04912	11.21	18.00	0.6227
	0.44235	100.12		

Molecular proportions:

$$0.9255 : 0.3084 : 0.6227 = 3 : 1 : 2 = 3\text{ZnO} \cdot \text{CO}_2 \cdot 2\text{H}_2\text{O}.$$

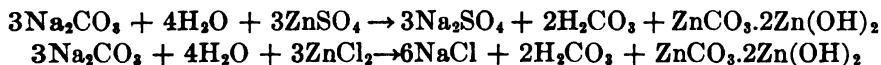
The precipitate from the reaction $\text{ZnCl}_2 + \text{Na}_2\text{CO}_3$:

Chemical Components	Grams	Per Cent.	Molecular Weight	Molecular Ratio
ZnO.....	0.32560	75.33	81.40	0.9252
CO ₂	0.05866	13.59	44.00	0.3088
H ₂ O.....	0.04799	11.10	18.00	0.6166
	0.43225	100.02		

Molecular proportions:

$$0.9252 : 0.3088 : 0.6166 = 3 : 1 : 2 = 3\text{ZnO} \cdot \text{CO}_2 \cdot 2\text{H}_2\text{O}.$$

The above results clearly show that each of these precipitates represents, not a normal zinc carbonate, but a basic zinc carbonate such as hydrozincite. Its reaction may be represented by some such equation as this:



Moreover, Dana, in his *System of Mineralogy*, suggested this formula for hydrozincite. According to him, the constituents of a hydrous zinc carbonate from Auronz, were as follows: ZnO, 73.21; CO₂, 14.55; and H₂O, 11.38 per cent.

In another experiment, the reactions between Na_2CO_3 and ZnSO_4 solutions gave likewise a precipitate of basic zinc carbonate, but it was found to contain only one molecule each of ZnCO_3 and $\text{Zn}(\text{OH})_2$. This difference probably indicates that the formation of basic zinc carbonate varies in composition according to conditions.²⁴

²⁴ Alex. Smith. *General Inorganic Chemistry*, p. 649.

The precipitate obtained from the reaction of potassium carbonate with ZnSO_4 or ZnCl_2 was identical in appearance, but no analysis was made.

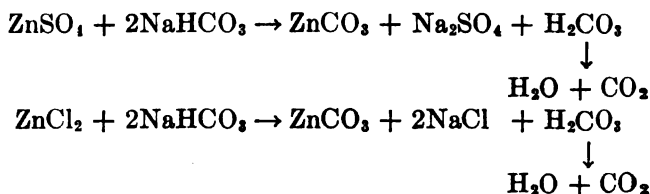
Duplicates of this experiment between ZnSO_4 and Na_2CO_3 were performed at the same time and under same conditions; but instead of filtering off the precipitate, carbon dioxide was allowed to pass directly into the mixture for half an hour to one hour. The mixture (formerly precipitate and liquid) became in this time a clear solution showing that ZnCO_3 is not stable in presence of free H_2CO_3 . This agrees with the fact that in the waters of zinc mines, where zinc carbonates occur, no H_2CO_3 has been reported.

(b) 1. Bicarbonate of Sodium (NaHCO_3): To an excess of this bicarbonate solution was added 400 c.c. of $\frac{N}{50}$ ZnSO_4 solution. The mixture soon became milky and a very fine precipitate was formed. Some CO_2 gas appeared during the precipitation. In the same way ZnCl_2 was treated with NaHCO_3 and the same kind of precipitate was formed. Each mixture was heated almost to boiling in order to drive off the excess of CO_2 . Each precipitate was separated from its liquid by filtration, and was washed until no sign of SO_4 or Cl was shown. The precipitate in each case was amorphous.

The weight of the precipitate from ZnSO_4 was 0.5112 g., and that from ZnCl_2 , 0.4873 g. After being dried in the oven and cooled in the desiccator they yielded the results given below:

The precipitate from the reaction between ZnSO_4 and NaHCO_3			The precipitate from the reaction between ZnCl_2 and NaHCO_3		
Grams, Per Cent.			Grams, Per Cent.		
ZnO	0.3302 =	64.6	0.3164 =	64.94	
CO_2	0.1798 =	35.18	0.1712 =	35.14	
H_2O	trace		none		
Total.....	0.5100	99.78	Total.....	0.4876	100.08

The substance is evidently anhydrous carbonate of zinc and this reaction may be represented by the following simple equation:



Theoretically anhydrous carbonate of zinc contains: ZnO , 64.8; CO_2 , 35.2 per cent.

(b) 2. Bicarbonate of Calcium: 1.001 g. of finely powdered pure limestone (an equivalent of $\frac{N}{50}$) were placed in 1,000 c.c. water, into which carbon dioxide was subsequently allowed to pass until the powdered mineral entirely disappeared, being converted into the more soluble bicarbonate. The reaction may be represented by the following equation:



400 c.c. of this lime solution were mixed with a slight excess of $\frac{N}{50}$ solution of ZnSO_4 in one flask and ZnCl_2 in the other. Each mixture immediately became milky and a very fine precipitate formed. The liquid was almost neutral. After being heated to boiling each precipitate was filtered off, washed until the last few drops of washing water no longer showed the presence of SO_4 or CO_3 ions. Then each was dried in the steam oven. They were weighed, after they had been allowed to cool in the desiccator, and yielded respectively:

A. The precipitate from $\text{Ca}(\text{HCO}_3)_2 + \text{ZnSO}_4$ weighed 0.4581 g.

	Grams	Per Cent.	Molecular Weight	Molecular Ratio	Molecular Proportion
CaO	0.11290	24.63	56.1	0.4389	1
CO_2	0.17862	38.98	44.0	0.8700	2
ZnO	0.10650	36.34	81.4	0.4287	1

B. The precipitate from $\text{Ca}(\text{HCO}_3)_2 + \text{ZnCl}_2$ weighed 0.4272 g.

	Grams	Per Cent.	Molecular Weight	Molecular Ratio	Molecular Proportion
CaO	0.11070	25.90	56.1	0.4616	1
CO_2	0.10743	39.19	44.0	0.8906	2
ZnO	0.14910	34.90	81.4	0.4287	1

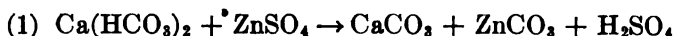
Their mineral components according to the amount of the above chemical analyses, may be represented respectively:

(A)	(B)
$\text{CaO} \cdot \text{CO}_2$ or $\text{CaCO}_3 = 0.2014$ g.	$\text{CaCO} \cdot \text{CO}_2$ or $\text{CaCO}_3 = 0.1970$ g.
$\text{ZnO} \cdot \text{CO}_2$ or $\text{ZnCO}_3 = 0.2566$ g.	$\text{ZnO} \cdot \text{CO}_2$ or $\text{ZnCO}_3 = 0.2320$ g.

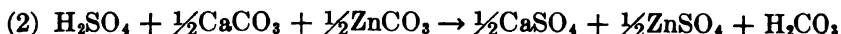
From this it appears that the compound consists of carbonates of calcium and zinc. From the amount of calcium carbonate used at the

start, before its conversion of $\text{Ca}(\text{HCO}_3)_2$, and from the amount of the mineral component of the same compound, it is readily seen that the amount of Ca present in 400 c.c. of $\frac{N}{50}$ $\text{Ca}(\text{HCO}_3)_2$ solution is 0.1614 g. and that the metal recovered in its original form of CaCO_3 by interaction with ZnSO_4 is 0.0808 g. or a little over half of its original amount.

It is also theoretically known that one molecule of $\text{Ca}(\text{HCO}_3)_2$ in solution is able to precipitate an equivalent of ZnCO_3 , and to produce a complete reformation of CaCO_3 , provided they are not affected by the byproduct H_2SO_4 , as indicated in the equation:



But this hardly appears to be correct, as shown by the amount of the mineral components which is only about half of what this equation demands. Suffice it to say that the formation of the "secondary" H_2SO_4 (or HCl , if ZnCl_2 is used instead of ZnSO_4) must be used up in its reaction with the chemically precipitated ZnCO_3 , and CaCO_3 , thus converting part of both of these precipitates into ZnSO_4 and CaSO_4 on the one hand, and giving rise to the formation of H_2CO_3 on the other until the new equilibrium is reached. These relations may be best shown by the following equation:



As soon as this condition is reached the rest of the CaCO_3 and ZnCO_3 will be able to remain as precipitates, since these carbonates are quite stable in the solution of their sulphates. It is also possible that the carbonates of Zn and Ca may co-exist as one compound, analogous to dolomite, whose formula is ordinarily written: $\text{MgCO}_3.\text{CaCO}_3$.

The H_2CO_3 shown in equation (2) doubtless breaks up into H_2O and CO_2 . These relations are further proved by the qualitative test of the filtrate which, after being heated to boiling for 15 min., showed only the presence of Ca, Zn and SO_4 ions.

These reactions may help to account for the occurrence of zinc carbonate in, and in contact with, limestone in nature, as in the Bertha Mines of Virginia.

Dr. Watson²⁵ has suggested that the calcium carbonate must be first converted into soluble calcium bicarbonate due to "the interaction of the percolating atmospheric waters containing carbon dioxide in solution on limestone," before the reaction to form zinc carbonate takes place. This is in full accord with the writer's experience in the laboratory. He further suggested that in the formation of normal zinc carbonate, under natural conditions, the compound is probably in the form of basic carbon-

²⁵ T. L. Watson. *Bulletin No. 1, Geological Survey of Virginia*, p. 44 (1905).

ate as an "intermediate step" towards the formation of normal zinc carbonate from the reactions between zinc sulphate and calcium bicarbonate. But the writer's laboratory experience tends to indicate that normal zinc carbonate only is formed if the reagent is a bicarbonate, whereas a basic carbonate of zinc only is precipitated in varying composition if the reagent is a normal carbonate.

The precipitates from duplicates were not filtered, but directly treated with carbon dioxide, which was allowed to pass into the mixture for about three-quarters of an hour. At the end of this period the precipitates had entirely disappeared and the liquid had become clear, indicating that in presence of free H_2CO_3 the normal carbonate of zinc cannot exist, probably just as the calcium carbonate became soluble, forming bicarbonate, under the same condition, as shown in the preparation of it for the above experiments.

Similar experiments were performed with chemically made sulphide of zinc and with natural sulphide from Joplin, Mo. In these experiments 0.4873 g. (the equivalent of $\frac{N}{100}$) of each of these sulphides was treated in 1,000 c.c. water and mixed with a double amount of $\frac{N}{100}$ solution of normal sodium carbonate in one flask, and of bicarbonates of sodium and calcium in the other, as before. No evidence of appreciable change in each case was noted in the course of two months. Carbon dioxide was then passed into each of these mixtures for an hour, but evidence of change was equally absent in each case. This suggests that direct oxidation of sphalerite to zinc carbonate is far from being realized under laboratory conditions.

Experiment 6a.—On the Conversion of Zinc Carbonate from Zinc Sulphide. These experiments were repeated, but only the natural sulphide of zinc was used, with modifications in the conduct of the experiments. 100 g. of coarsely crushed (not powdered) sphalerite was spread in layers in a large glass jar (A, Fig. 3), 9 in. high and 4 in. in diameter, with alternate layers of various sizes of sandstones, shales and limestones above and below the sulphide layers. The entire charge filled the jar three-quarters full. A solution of $\frac{N}{50} \text{CuSO}_4 \cdot 5\text{H}_2\text{O}$ was first allowed to flow slowly into this jar through a glass tube (T_1) and as soon as the solution filled the jar up to three-quarters of the depth, the solution was allowed to flow through another glass tube (T_2), which was fixed to reach downward one-fourth of the depth of the jar, into another bottle which served as a receiver (C) containing the excess of the solution of CuSO_4 , which was returned to the supplying bottle (B) from time to time. Altogether 18,000 c.c. of $\frac{N}{50} \text{CuSO}_4$ solutions were used during the entire period, including fresh portions of this solution, added from time to time

to compensate for evaporation. This treatment was continued for a period of four months, under ordinary conditions of temperature and pressure.

At the end of this period the colorless solution and the blackish precipitate, formed during the operation in the jar (A), and the solutions in bottles (B) and (C), amounting by this time to about 3,400 c.c., were all emptied into a large flask. The sandstone, shale and limestone together

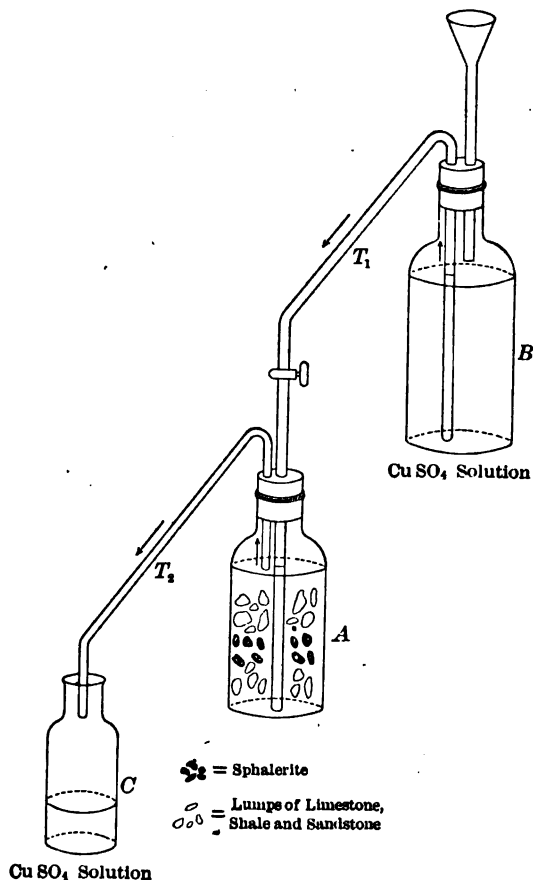


FIG. 3.—APPARATUS FOR EXPERIMENT 6a.

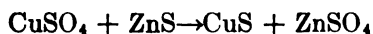
with the remaining lumps of sphalerite in jar (A), were carefully washed with water until the washing water was free from SO_4 ions. All the washings were added to the mixture.

The blackish precipitate was then filtered off from this mixture and washed until the washings showed no presence of SO_4 ions. These washings were also added to the filtrate which was then made up to 5,000 c.c. with distilled water.

It was the purpose of this experiment to ascertain in a quantitative way: (1) whether copper as sulphate in dilute solutions would attack the natural ZnS in contact with limestone, shale and sandstone; and (2) whether such reagent would also dissolve part of those rocks in the jar, apart from sphalerite, in the course of circulation. To determine these two points, definite portions of the filtrate and the precipitate, whose weight after being dried in the desiccator was found to be 17.2 g., and which contained some free sulphur, were taken for analysis. The filtrate, according to litmus paper tests, was slightly acidic or almost neutral; 500 c.c. of it were used in the analysis recorded below: The precipitate yielded:

				Amount in Precipitate and in Filtrate	Amount of Cu and SO ₄ Originally Present in Solution
ZnO.....	2.1790	Zn	1.7520	1.7520	
Cu.....	11.4400	Cu	11.4400	11.4400	11.4400
CO ₂	1.1790				
S including free S..	5.7600	gangue	7.4662	7.4662	
SiO ₂	0.1002				
	20.6582		20.6582		
and from the fil- trate were ob- tained					
SiO ₂	0.0018	SiO ₂	0.0018		
Al ₂ O ₃	0.00366	Al	0.00194		
FeO.....					
Fe ₂ O ₃	0.00844	Fe	0.00590		
CaO.....	0.03673	Ca	0.02625		
MgO.....	0.06253	Mg	0.03775		
Na ₂ O.....	0.00824	Na	0.00612		
K ₂ O.....	0.01250	K	0.01038		
SO ₄	1.72810	SO ₄	1.72820	17.2820	17.2820
ZnO.....	1.24700	Zn	1.00200	10.0200	
				47.9602	28.722

From the molecular weight, it is known that 18,000 c.c. $\frac{N}{50}$ CuSO₄·5H₂O solution furnish about 11.448 g. Cu, and from the chemical reactions,



the amount of copper (11.440 g.) actually found in the precipitate would require, by calculation, about 5.76 g. sulphur to form CuS.²⁶ The amount

²⁶ The CuS is believed to have been reduced to Cu₂S and the free S in the precipitate is due to this reduction.

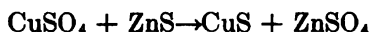
of zinc (1.7520 g.) was undoubtedly in the form of carbonate, precipitated from zinc sulphate, as may be inferred also from the presence and amount of carbon dioxide in the analysis. Thus by calculation, according to the above analysis, the precipitate contains the following:

	Grams		Grams
CuS	= 17.20	= Cu	11.4400
ZnCO ₃	= 3.358	= Zn	1.7520
SiO ₂	= 0.1002	= gangue	7.4662
	20.6582		20.6582

The analysis of the filtrate, through the absence of Cu, points to its precipitation from solution, and the presence of an almost equivalent amount, molecularly speaking, of zinc in the solution as sulphate, indicates the oxidation of zinc from its sulphide. Lastly, the presence of other substances, which were absent in the mixture at the beginning of the experiment, indicates that a solution containing sulphate of copper (or a soluble salt of any other metal beneath hydrogen in the electromotive-force series) tends to carry many common rock-making minerals or components into solution and, as a consequence, neutralize or decrease its acidic character to a very appreciable extent.

It is also known from the molecular weight that 18,000 c.c. ^N₅₀ solution of CuSO₄ furnish 11.448 g. Cu and 17.2818 g. SO₄.²⁷ Theoretically, a ^N₅₀ solution of cupric sulphate in one liter requires 0.654 g. Zn to displace all the Cu.

But from the fact that Cu is entirely absent in the filtrate, as shown by the above analysis, the theoretical amount of Zn oxidized into sulphate or required to replace the amount of Cu actually found in the precipitate, cannot be less than 11.76 g., as based on the reaction,



By deducting 11.76 g. (to satisfy the replacement of the Cu) from the amount of Zn found in the filtrate (10.02 g.) and in the precipitate (1.752 g.) there is an excess of 0.012 g. (11.772 - 11.76 = 0.012), which is about 0.1019 per cent. more than what is theoretically necessary for the amount of copper as found in the analysis of the precipitate. The difference in these values (between theoretical and practical calculations) is undoubtedly due to errors in manipulation.

The amount of SO₄ in the original solution as represented by CuSO₄ at start was 17.2818 g. (theoretically).²⁷ The amount recovered was

²⁷ In calculating the SO₄ from the sulphate of copper no water of crystallization was considered.

17.282 g. After deducting from the total amount of SO_4 (17.282 g.) the theoretical amount required for all the Zn as sulphate found in the filtrate, there remains 2.552 g. ($17.282 - 14.73 = 2.552$) which may be safely assigned to the other bi- and tri-metals found in the filtrate.

That the sulphide of zinc is oxidized by the solution of copper sulphate is practically certain. The remaining solution and the washings, chiefly containing Zn as sulphate, are now ready for the treatment as used in experiment 6b.

Experiment 6b.—The precipitation of ZnCO_3 from the oxidized solution of ZnS : 500 c.c. of this mixed solution was treated with an excess of $\frac{N}{50}$ solution of NaHCO_3 . A very fine precipitate was formed immediately. The whole mixture was gently heated almost to boiling; some CO_2 was noted passing off during the heating. The mixture was allowed to cool and the precipitate was separated from the liquid by filtration. The former was washed until the washings showed no sign of the presence of SO_4 . It was then dried in the steam oven and cooled in the desiccator before the precipitate was weighed (1.9884 g.) and analyzed. The following are the analytical results:

	Gram	Per Cent.
SiO_2	0.0019	0.095
CaO	0.0367 (= Ca, 0.02625)	1.842
MgO	trace	
H_2O	none	
CO_2	0.7105	35.260
ZnO	1.2470 (= Zn, 1.002)	62.560
	1.98838	99.757

From the above analysis, it may be safely assumed that all the chemical components present (except SiO_2) were in the form of carbonates, since, by recasting for their carbonates it is found, as shown below, that the amount of CO_2 required by these by-metals (Zn and Ca) is 0.7028 g. Namely,

0.02625 g. Ca will form 0.0655 g. CaCO_3 with 0.0288 g. CO_2

1.0020 g. Zn will form 1.9210 g. ZnCO_3 with 0.6740 g. CO_2

0.7028 g.

The remaining CO_2 (0.0077 g.) is probably in combination with MgO ; but the latter is too small in amount to be considered. The amount of these mineral components as carbonates is obtained by calculation from their chemical components. Their mineral component proportion may be represented as follows:

	Grams	Per Cent.
CaCO ₃	0.0655	3.294
MgCO ₃	trace	
ZnCO ₃	1.9210	96.61
		99.904

Since there is enough CO₂ to take care of all the zinc found, and no water is present, and SiO₂ is too small in amount to be considered as having combined with Zn in any way, the Zn compound is, beyond doubt, a normal carbonate.

In another experiment a mixture of dilute solutions of bicarbonate was allowed to pass through a jar containing sphalerite in the manner used for the experiments under 6a. The operation was continued for two months, but met with no success.

In the light of the above results, it seems quite certain that ZnCO₃ is not formed direct from its sulphide. The sulphide is first oxidized into zinc sulphate, and the latter may migrate a short or long distance, as the case may be, before the carbonation takes place. In the waters which percolate through strata in which ZnS (sphalerite) occurs, the chemically active compounds must be complex, and must be provided with oxidizing material of one kind or another. These waters will doubtless oxidize the sphalerite at an incipient stage to sulphate. The oxidized part will, according to existing conditions, probably be transported to some other place until it meets an alkaline solution or solutions of alkaline earths, which in nature mostly contains soluble carbonates. The deposition of ZnCO₃ will then take place, if the metals like Pb or Cu, which will form more insoluble carbonates than Zn, are either absent or very small in amount. These conditions will account for the fact that both ZnCO₃ and ZnS have been found in different horizons as if they had no relation to each other, as in the Circle Mines at Oronogo, Mo.²⁸ and in some zinc mines in Virginia.²⁹

In case an orebody of sphalerite is attacked by a neutral solution of sulphates and carbonates, the sphalerite may be first oxidized into sulphate, especially when the metals above zinc in Schuermann's series³⁰ are present as sulphates in the solution, and then immediately carbonated, without suffering the removal of any appreciable quantity by migration. This may help to account for the fact that sphalerite is sometimes closely associated with smithsonite, and the latter always grades downward into the former.

4. The Reaction of Zinc Sulphate Solution with Limestones.—The

²⁸ The writer had an opportunity in the summer of 1913 to visit some of the zinc mines in southwest Missouri, particularly in the Joplin district. All the writer's own field data were obtained at that time.

²⁹ T. L. Watson. *Bulletin No. 1, Geological Survey of Virginia*, p. 45 (1905).

³⁰ *Liebig's Annalen der Chemie*, vol. cexlvii, p. 326 (1888).

*Experiment 7: With Crystalline Limestone**

Analysis of the Limestone					Analysis of the Solution	
Before Treatment			After Treatment†		Before Treatment	After Treatment
	Grams	Per Cent.	Grams	Per Cent.	Grams	Grams
SiO ₂	0.0427	0.6521			CaO	0.6063 = 1.472 CaSO ₄
Al ₂ O ₃	0.0234	0.3574			SO ₄ 2.597	2.5967
Fe ₂ O ₃	0.0183	0.2796			ZnO	1.3200 = 2.619 ZnSO ₄ or
CaO.....	3.543	54.1200 = 96.54	2.9380	43.1200 = 76.93	Zn 1.768	1.0470
		CaCO ₃		CaCO ₃		
MgO.....	0.0193	0.2947 = 0.6155	0.0156	0.2290 = 0.4769		
		MgCO ₃		MgCO ₃		
CO ₂	2.800	42.7408	2.7966	41.0400		
ZnO.....			0.8800	12.9100 = 19.9000		
			or	ZnCO ₃		
Zn.....			0.7071			
	6.4467	97.1655		97.3069		

Experiment 8: With Impure Limestone

Analysis of the Limestone					Analysis of the Solution	
Before Treatment			After Treatment		Before Treatment	After Treatment
	Grams	Per Cent.	Grams	Per Cent.	Grams	Grams
Insoluble and organic matter.....	3.432				CaO	1.16
SiO ₂	2.406				MgO	0.2813
Al ₂ O ₃	3.512				SO ₄ 2.597	2.597
Fe ₂ O ₃	0.9223				ZnO 1.768	
CaO.....	13.5200	34.26 = 61.12	12.380	31.37 = 55.94		
		CaCO ₃		CaCO ₃		
MgO.....	1.9680	4.987 = 10.42	1.687	4.275 = 8.933		
		MgCO ₃		MgCO ₃		
H ₂ O + ...	0.3117					
H ₂ O -	0.1947					
CO ₂	12.751		12.735			
MnO.....	trace					
Na ₂ O.....	0.1270					
K ₂ O.....	0.325					
ZnO.....			2.201	7.158 = 11.03		
Zn.....			1.750	ZnCO ₃		
	34.4697	71.54				

* This limestone was obtained by Dr. C. P. Berkey from Rockland. Dr. Berkey gave CaCO₃, 96.45, and MgCO₃, 0.12 per cent. in one case; and CaCO₃, 94.9 and MgCO₃, 20 per cent. in the other.

† H₂O and organic matter was not determined. Only carbonates determined.

solution of zinc sulphate used in the following experiments is all ^{N,}₂₀ that is, 4.365 g. ZnSO_4 or 1.768 g. metallic zinc in every 1,000 c.c. The limestones used were analyzed before and after the treatment, which in each case was kept going for a period of four months in a closed bottle, to which air was occasionally admitted through a side tube. In each case a definite weight of limestone was treated with 1,000 c.c. ^N₂₀ solution of ZnSO_4 . At the end of four months the limestone in each case was found to have been covered with a white thin coat which, upon drying, appeared to be somewhat powdery, and which was ZnCO_3 as shown in the respective analyses recorded on preceding page.

From the above analyses, it is shown that in the case of crystalline limestone, only two-thirds of the zinc was precipitated as ZnCO_3 , while with the impure limestone, zinc is entirely absent in the sulphate solution after the treatment. These apparent differences from the treatment of two different limestones with the same solution of ZnSO_4 are believed to be due to the presence of organic matter, which is common in impure limestone.

It is further shown that carbonates of both magnesium and calcium can be replaced by zinc under the conditions described above.

In another experiment with the impure limestone, minute particles of what appeared to be ZnS could be seen with the aid of a powerful lens. Such precipitation of the zinc as sulphide, by commingling of the oxidizing and reducing solutions in limestones, is not only possible, but also probable, according to some geologists, notably Dr. Buckley,³¹ in his studies of the origin of zinc sulphide in the Joplin district. The evidence that these minute particles were ZnS , was, however, not conclusive, for the reason that it was difficult to separate them from the whitish coating of zinc carbonate. An attempt was made to prepare thin sections, but the replaced material was too soft for grinding.

C. *Peculiar Conditions Affecting the Formation of Zinc Silicate (Calamine) and Zinc Carbonate*

5a. *The Effect of Carbon Dioxide upon the Basic Silicate and the Carbonates of Zinc in Water, and its Bearing upon the Conditions of their Precipitation.*—Hydrous silicate of zinc³² has long been known to be soluble in water in presence of CO_2 gas, but no one has attempted to explain in the light of modern chemistry the bearing of the latter upon the formation of this silicate and the association of the silicate with carbonate of the

³¹ E. R. Buckley. *Types of Ore Deposits*, pp. 103 to 132.

³² In 1849, Victor Monheim made his quantitative determination of zinc silicate by dissolving it in water in presence of carbon dioxide. *Verhandlungen des Naturhistorischen Vereins*, vol. vi, pp. 1 to 24.

same metal. The following experiments were undertaken in this connection.

Experiment 9a.—5 g. of pure calamine from the vicinity of Chihuahua, Mexico, secured from the collection of ores in the Department of Geology at Columbia University, were powdered until they passed 100-mesh sieve. This amount was mixed with 1 liter of water in a 2-liter flask. A stream of carbon dioxide was allowed to pass through for 80 hr., with occasional shaking of the contents. The mixture was filtered and a residue collected and washed before it was dried in the oven. The washings were added to the filtrate, which was perfectly clear. After being dried in the oven and cooled in the desiccator, the residue was weighed at 3.48 g. For convenience it is designated as first residue.

The filtrate was heated to boiling for about half an hour in order to drive off the CO_2 completely. A fine precipitate came down, which was also filtered, washed and dried in the oven. Its weight was found to be 1.462 g. It is designated as second residue. There is a loss of 0.012 per cent. due to manipulation.

By analysis these residues yielded respectively:

First Residue	Grams	Per Cent.	Second Residue Gram	Residue Per Cent.
ZnO.....	2.3480	67.48	0.9868	67.50
SiO ₂	0.8714	25.03	0.3661	25.03
H ₂ O.....	0.2597	7.483	0.1091	7.461
		99.973		99.991

That the hydrated silicate of zinc is soluble in water in presence of enough carbon dioxide, and that the same silicate can be reconverted into its solid form from such solution by raising the temperature, there is no doubt. It is also believed that the carbon dioxide does not combine with zinc to form some such soluble compound as bicarbonate of zinc, for the H_2O in the zinc silicate is basic,³³ and if bicarbonate were formed, zinc carbonate would come down as precipitate when the CO_2 was driven off by heating. Thus the carbon dioxide merely exerts its influence as a catalytic agent in carrying the silicate of zinc into solution, without affecting any change in its molecular combination.

Experiment 9b.—The next question is whether or not an impure calamine, that is a mixture of zinc silicate and zinc carbonate with carbonates of some other bi-metals will also become soluble in water in excess of carbon dioxide as shown in each individual case. For this purpose a piece of impure calamine, obtained from a miner in Joplin, Mo., was chosen for the experiment.

In order to know the exact contents, 3.259 g. of the powdered material from the specimen were analyzed, with the following results:

³³ Dana. *System of Mineralogy*, Sixth Edition, p. 548.

Chemical Components			Mineral Components		
	Grams	Per Cent.		Grams	Per Cent.
SiO ₂	0.5622	17.300	Calamine	2.2470	69.130
Fe ₂ O ₃	0.0377	1.160	Zinc carbonate	0.7632	23.480
FeO.....	trace		Calcium carbonate	0.1378	
CaO.....	0.0772	2.375	Magnesium carbonate.....	0.0786	4.240
MgO.....	0.0376	1.157	Limonite.....	0.0443	2.418
CO ₂	0.3673	11.310			1.363
ZnO.....	2.0110	61.870			
H ₂ O.....	0.1758	5.409			
Na&K.....	trace				
MnO.....	trace				
P ₂ O ₅	none				
	3.2688	100.581		3.2709	100.631

An equal amount (3.259 g.) of this material was taken for the treatment in the experiment, which was in every respect the same as in the last case. At the end of the experiment, the mixture was filtered. Upon boiling the filtrate, as before, a heavy precipitate collected. After being dried in the oven and cooled in the desiccator, it was found to weigh 2.522 g. and yielded:

Chemical Components				Mineral Components			
	Grams	Per Cent.	Per Cent.		Grams	Per Cent.	Per Cent.
SiO ₂	0.4109	16.2900	12.6400	Calamine.....	1.6412	65.0700	50.4900
Fe ₂ O ₃	0.0107	0.4243	1.2650	Zinc carbonate.	0.7311	28.9900	22.5000
FeO							
CaO.....	0.0411	1.6290	1.0490	CaCO ₃	0.0734	2.9110	2.2500
MgO.....	0.0341	1.3520	0.3293	MgCO ₃	0.0713	2.8280	2.1940
CO ₂	0.3261	13.2000	10.0300	Limonite.....	0.0126	0.4997	1.3377
ZnO.....	1.5825	63.2800	48.7000				
H ₂ O.....	0.1243	4.9280	3.8230				
	2.5297	99.1033	77.8363		2.5296	100.2987	77.8297

The chemical components were calculated, by recasting, into mineral components for the purpose of finding the exact amount of each mineral component dissolved in water in presence of the carbon dioxide. Its percentage was then recalculated (column 3) on the basis of the original weight of the material used in the experiment.

By comparing the figures in column 3 with the mineral percentage in the analysis of the specimen it can be readily seen: (1) that in the presence of carbon dioxide, nearly all the carbonates in the specimen were carried

into solution; (2) that the zinc carbonate was almost entirely dissolved; and (3) that a much greater percentage of zinc silicate was dissolved from the impure specimen than from the pure one used in experiment 7a. It is, therefore, plausible that the presence of the impurities in the specimen, particularly ZnCO_3 , must have exerted some influence upon the solution of the zinc silicate. It is at least suggestive that the molecules of ZnCO_3 , which were present with those of the hydrous silicate of zinc in the material used, would attract molecule by molecule the components of the silicate in water under the influence of carbon dioxide, but would not necessarily effect any change in their respective molecules or molar-volumes, since their equilibrium remains the same in the aqueous state under the above conditions.

5b. *The Reactions of Solutions of Soluble Silicate with Calamine and Smithsonite in Solution in Presence of Carbon Dioxide.*—Since the excess of carbon dioxide can hold both silicate and carbonate of zinc in solution, the question arises as to what peculiar effect will be produced when a silicate solution is introduced into such a mixture. That is to say, whether the basic silicate can be brought about by means of a soluble silicate solution from a solution of zinc carbonate and zinc silicate made by dissolving the latter two under the influence of carbon dioxide.

It has long been known that silicon dioxide may replace carbon dioxide when heated with an alkaline carbonate,³⁴ and inasmuch as this property of silicon dioxide may have direct bearing on the origin of zinc silicate in association with zinc carbonate such as frequently found in zinc mines, the study of its precipitating capacity was undertaken.

Experiment 10.—4 g. of finely powdered calamine obtained from Chihuahua, Mexico, and 4g. of chemically pure zinc carbonate were mixed together. The mixture (8 g. altogether) was then digested in 1,000 c.c. water under the influence of carbon dioxide, which was allowed to pass slowly (one bubble for every three seconds) through the mixture for a period of 50 hr. At the end of this period, the mixture was filtered and a residue, which for clearness, will be designated as residue X, was removed. It was dried in the steam oven and weighed 3.9692 g. By analysis the residue X, yielded:

	Grams	Per Cent.	Molecular Weight	Molecular Ratio	Molecular Proportion
SiO_2	0.5380	13.550	60.4	0.2244	3
CO_2	0.6387	16.090	44.0	0.3656	
H_2O	0.1603	4.033	18.0	0.2241	
ZnO	0.6310	66.300	81.4	0.8145	3
	3.9680	99.973			

³⁴ Alex. Smith. *General Inorganic Chemistry*, p. 519.

The filtrate, after the removal of the residue X, was treated with an excess of dilute solution of Na_2SiO_3 . A gelatinous precipitate, designated as precipitate Y, immediately collected and was filtered off, dried in the steam oven, and weighed, after being cooled in the desiccator, 2.3763 g.

As shown by the analytical results recorded below, it is certain that the gelatinous precipitate is not calamine ($2\text{ZnO} \cdot \text{SiO}_2 \cdot \text{H}_2\text{O}$), but is a different substance, which according to the molecular proportions of the chemical components, as shown below, may be called zinc meta-silicate (ZnSiO_3 or $\text{ZnO} \cdot \text{SiO}_2$). The following is the analysis of the precipitate Y:

	Grams	Per Cent.	Molecular Weight	Molecular Ratio	Molecular Proportion
SiO_2	1.0120	42.600	60.4	0.705300	1
H_2O	0.0003	0.012	18.0	0.000667	0
ZnO^*	1.3610	57.280	81.4	0.703800	1
	2.3733	99.892			

$$7038 : 0.7053 : = 1 : 1 = \text{ZnO} : \text{SiO}_2 = \text{ZnSiO}_3.$$

The filtrate from the precipitate Y was colloidal. It was heated to boiling for half an hour until CO_2 was no longer given off. During the heating, a precipitate collected as expected. After being cooled, the precipitate was removed by filtration from the colloidal solution and was washed with distilled water. It was dried in the oven and cooled in the desiccator, and weighed 1.9747 g. For clearness, it will be designated as precipitate Z, from which were obtained:

	Grams	Per Cent.	Molecular Weight	Molecular Ratio	Molecular Proportion
SiO_2	0.4634	23.470	60.4	0.3886	1
H_2O	0.1381	6.993	18.0	0.3885	1
ZnO	1.2490	63.240	81.4	0.7769	2
A gelatinous residue	0.1235	6.253			
	1.9740	99.956			

$$0.7769 : 0.3886 : 0.3885 = 2 : 1 : 1 = 2\text{ZnO} \cdot \text{SiO}_2 \cdot \text{H}_2\text{O}.$$

From the molecular proportions of X, it can be readily seen that the residue consists of silicate and carbonate of zinc, as the undissolved portions of their original amount, or the proportion of the oxide of zinc is almost equal to the sum of those of the other three components, namely

* Zn in this case was determined as pyrophosphate and ZnO was calculated from the molecular formula of $\text{Zn}_2\text{P}_2\text{O}_7$ and found to contain 1.082 grams metallic zinc.

$0.2244 + 3.656 + 0.2241 = 8145 - 4$ (to be exact) or the amount of SiO_2 , CO_2 , and $\text{H}_2\text{O} =$ the amount of ZnO .

Thus $3\text{ZnO} = \text{SiO}_2 + \text{CO}_2 + \text{H}_2\text{O}$ or



On the basis of the original amounts used at the beginning of the experiment and by means of their molecular proportions, it is found that the residue X constitutes 2.151 g. or 53.966 per cent. of hydrous silicate of zinc and 1.82 g. or 45.5 per cent. of the normal carbonate of zinc as representing the undissolved portions from their original amounts; that from the precipitate (Z) were recovered 1.851 g. or 46.27 per cent. the original hydrous silicate of zinc. In other words, no change has taken place in the hydrous silicate of zinc, so far as its amount and chemical combination are concerned, whereas the amount of zinc carbonate (its original weight is 4 g.) left in the residue X is only 1.82 g. or 45.5 per cent. of the original amount. Its remainder (theoretically 2.18 g. or 54.5 per cent.) has undoubtedly contributed to the precipitation of the meta-silicate of zinc, the precipitate Y, as will be shown in the following:

	Grams
The original weight of ZnCO_3	4.000
The amount left in residue X.....	1.8200
The amount lost.....	2.1800

According to its molecular weight, 2.18 g. ZnCO_3 contain 1.137 g. metallic zinc, and if this theoretical amount had all been precipitated out as zinc meta-silicate, there would be 2.464 g. ZnSiO_3 as against the actual value of 2.376 g. ($2.3763 - 0.0003 = 2.376$). The difference between the two is about 3.5 per cent., which is evidently lost in the course of manipulation.

From these results, it is clear that zinc carbonate may be replaced by a solution of soluble silicate to form zinc silicate, but empirically, this compound in no way resembles calamine, since the water is not present in sufficient amount. The water, in the case of true calamine, would constitute one-half of the molecular proportion of the oxide of zinc as required by the formula, $2\text{ZnO} \cdot \text{SiO}_2 \cdot \text{H}_2\text{O}$.

It is also, however, a recognized fact that the water contained in calamine is basic and the mineral is probably a basic meta-silicate of zinc. For this reason, its formula may be represented by $\text{Zn}(\text{OH})_2 \cdot \text{ZnSiO}_3$ in place of $2\text{ZnO} \cdot \text{SiO}_2 \cdot \text{H}_2\text{O}$. (See the experiment under 6.)

6. *The Reactions of a Mixture of Hydroxide and Silicate of Sodium with a Dilute Solution of Zinc Sulphate.*—From the above suggestions it may be readily inferred that a mixture of Na_2SiO_3 and NaOH will precipitate some such compound, partly as zinc hydroxide, and partly as meta-silicate, from a dilute solution of a zinc salt. The following experiment

was designed in an attempt to demonstrate these suggestive properties of the mineral under discussion.

Experiment 11.—2,000 c.c. of very dilute solution of ZnSO_4 (1.61 g. ZnSO_4 per liter) were treated with an equally dilute mixture of Na_2SiO_3 and NaOH in excess. A rather heavy flocculent precipitate gradually collected. The amount obtained was 2.412 g. and yielded:

Chemical Components Per Cent.		Mineral Components Per Cent.	
H_2O	7.463	ZnSiO_3	58.83
SiO_2	25.020	$\text{Zn}(\text{OH})_2$	41.21
Zn	54.180 = 67.47 ZnO		
	99.953		99.95

According to this analysis, the amount of zinc in the precipitate was 1.307 g., as against 1.308 g. originally present in 2,000 c.c. of ZnSO_4 solution. By recasting from its chemical components, it is found that there are 1.4174 g. as zinc meta-silicate (ZnSiO_3) and 0.994 g. as zinc hydroxide or 58.83 per cent. ZnSiO_3 and 41.21 per cent. $\text{Zn}(\text{OH})_2$. If we consider the precipitate obtained in this way as a single compound, the writer sees no objection to writing $\text{ZnSiO}_3 \cdot \text{Zn}(\text{OH})_2$ for the chemical composition of calamine as a makeup of one part hydroxide and one part meta-silicate, since neither of the formulæ, $2\text{ZnO} \cdot \text{SiO}_2 \cdot \text{H}_2\text{O}$ and $(\text{ZnOH})_2 \cdot \text{SiO}_2$, given in Dana's *Mineralogy*, has any greater claims to being representative. It is far from safe, however, to regard this amorphous precipitate, as true calamine, for the reason that, with the exception of its index of refraction (very close to 1.62) no optical confirmations could be obtained from it.

In spite of this lack of sufficient optical evidences, however, there is reason to believe that calamine in nature is probably built up by steps as meta-silicate and hydroxide in their incipient stages, through some natural influence or condition, producing a continuous alternation in the formation of this mineral, which we are not able to imitate in the chemical laboratory. Since even a very dilute solution of soluble silicate (Na_2SiO_3) and zinc salts always gives an amorphous precipitate at the ordinary temperature and pressure, and since amorphous calamine is not encountered in nature, the writer is constrained to believe that the natural calamine must have crystallized very slowly from very dilute siliceous solutions.

D. High Temperature Methods

7a. *The Influence of Temperature on the Formation of Calamine.*—*Experiment 12:* In connection with the preparation³⁵ of willemite accord-

³⁵ The writer desires to thank R. J. Moore of the Department of Chemistry at Columbia University, most heartily for placing at his disposal the apparatus without which it would have been impossible to conduct satisfactorily any of these experiments, and to C. J. Morrell for kind assistance.

ing to M. Gorgeu's method,³⁶ the writer succeeded in obtaining minute crystals of calamine instead, by treating a mixture containing one part of H_4SiO_4 and 30 parts of one equivalent of Na_2SO_4 and one-half equivalent of ZnSO_4 . In these experiments only anhydrous Na_2SO_4 and ZnSO_4 were used. Various amounts (from one-half to one equivalent) of the latter compound in different experiments were tried, but one-half equivalent of ZnSO_4 with one equivalent of Na_2SO_4 in 30 parts of these two compounds, is the suitable amount. All three compounds, well mixed in a porcelain crucible made for high-temperature work, were first heated over a Bunsen burner; and when the fusion began, the crucible was transferred to the "asbestos nest" built in an iron vessel, which was placed in the middle of an assay furnace previously heated. By so doing the temperature inside of the "asbestos nest" may be easily kept at or below 900°C . (this was measured by a LeChatelier thermo-couple pyrometer, protected by a long porcelain tube, which was inserted vertically into the "nest") in order to prevent any zinc from volatilization during the fusing, while the temperature outside the iron vessel is probably over $1,000^\circ\text{C}$. The heating in this was continued from one-half to one and one-half hours. If the porcelain crucible is transferred into the furnace without using the "asbestos nest," the results are comparatively poor and indicate the volatilization of zinc by the sudden increase of temperature and quick fusion. In this case the odor of SO_2 is quite noticeable near the furnace. On the other hand, it has been experimentally shown that, when the fusion is quite complete, if the "asbestos nest" is used, the temperature may be raised to $1,300^\circ\text{C}$. or $1,400^\circ\text{C}$. without noticeable volatilization of zinc.

After the melt had been kept in the furnace for half an hour or more, the fused mass, which usually became covered with a thin yellowish and sometimes greenish sheet, was placed directly in boiling water. This boiling water treatment causes solidification and usually gives a green surface to the solid mass, especially along the edge. If the mass has been heated too long in the furnace, the solution from the boiling water usually shows a basic reaction; if under-heated the reaction with litmus paper is neutral. By heating in the furnace from half to one hour in the "asbestos nest" as described above, we secure the best results.

After the fused mass, solidified in the boiling water, had been cooled, the contents were taken out with a knife and boiled in distilled water for a long time in order to remove any sulphates crystallized out on the sudden cooling. The desired silicate, if formed, would remain insoluble either in masses or in crystals. This is found to be true, but in each case the crystals of the silicate, as proved below, are very minute and are so small in amount that it is difficult to separate them from the greenish

³⁶ *Bulletin 10, Société, Française de Mineralogie*, pp. 30 to 39.



FIG. 4.—MICROPHOTOGRAPHS OF THE CRUSHED FRAGMENTS FROM THE SYNTHETIC CRYSTALS OF CALAMINE PRODUCED BY HEAT FROM A CHEMICAL MIXTURE.

No. 1, magnified 215 diameters, crossed nicols. No. 2, magnified 25 diameters, plain light.

fine residues which equally remain insoluble and which, by qualitative tests, gave SiO_2 , Zn, and water vapor.

These minute crystals of the silicate are different in shape and may be divided into two kinds, respectively white and light green in color. The white ones are more or less tabular in form and fibrous in structure, while the green ones seldom show any definite crystal outline. There is no difference in their chemical composition, but optically the white silicate is positive, whereas the green is negative though both give biaxial interference figures.

As suggested by Mr. Colony, Professor of Mineralogy and Geology in Cooper Union in the City of New York, the white crystals which are optically positive and biaxial may be regarded as *m* calamine; the green ones which are optically negative and biaxial, may be considered as *n* calamine. The latter probably represents a transitional stage in the formation of calamine proper. It is further thought that such variation is not uncommon in the behavior of many minerals. Experience shows that a mineral may under special conditions, change even its system as in case of leucite³⁷ which cannot remain isometric below 560°C.

The following are the results obtained from the microscopic examinations (see Fig. 4):

	White Crystal	Green Crystal
Shape.....	Elongated	Lath shaped and elongated
Color (transmitted light)	Light bluish	Light greenish
Indices of refraction.....	1.63	1.61
Extinction.....	Parallel	Parallel
Extinction angles.....	85°-90°	Mostly 90°
Birefringence.....	<0.04	>0.06
Elongation.....	+	-
Interference figure.....	Biaxial	Biaxial
Optical sign.....	+	-

The following are the chemical analyses of the respective silicates:

White				Green			
	Per Cent.	Molecular Weight	Molecular Ratio		Per Cent.	Molecular Weight	Molecular Ratio
H_2O	7.457	18.0	0.4143	H_2O	6.825	18.0	0.3792
SiO_2	24.990	60.4	0.4137	SiO_2	25.030	60.4	0.4144
ZnO	67.480	81.4	0.8289	ZnO	67.503	81.4	0.8292

The molecular proportion of water for the green silicates is about 0.82 per cent. lower than what is theoretically required, but this difference

³⁷ N. H. and A. N. Winchell. *Elements of Optical Mineralogy*, p. 275 (1909).

is so small that it may be safely attributed to the manipulation. Aside from this, either silicate has the following relation, viz. $\text{H}_2\text{O} = 1$, $\text{SiO}_2 = 1$, $\text{ZnO} = 2$. In other words these silicates or crystals have the formula $2\text{ZnO} \cdot \text{SiO}_2 \cdot \text{H}_2\text{O}$, or $(\text{ZnOH})_2\text{SiO}_3$ or $\text{Zn}(\text{OH}_2) \cdot \text{ZnSiO}_3$ (as suggested by the writer in an earlier paragraph). The question, which is the correct formula, remains to be determined, and is hardly within the province of this work.

From these chemical and microscopic results it is clear that the crystals obtained under the conditions described above, are hydrous silicate of

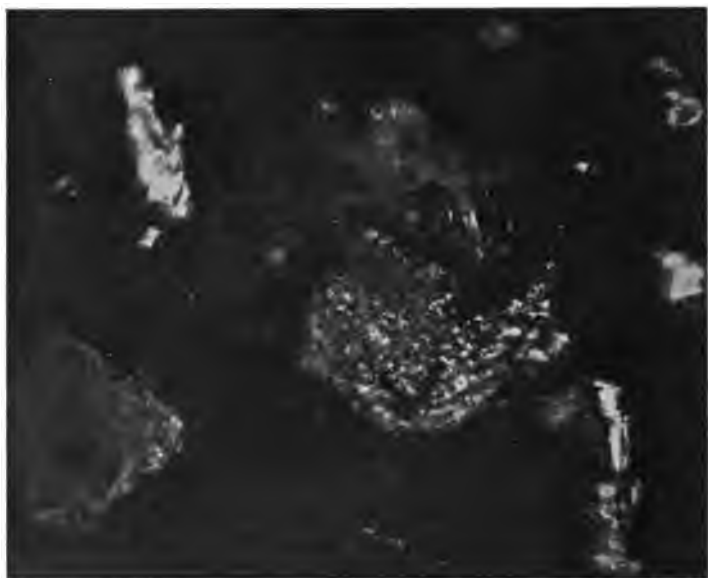


FIG. 5.—MICROPHOTOGRAPH OF THE CRUSHED FRAGMENTS FROM THE FUSED NATURAL CALAMINE.

These are some of the fragments showing the tendency of rearranging their molecules. Under high power some of the very minute hexagonal crystals in the fragments are well shown. This is no doubt due to the influence of heat, but these newly rearranged crystals are too minute to show on photograph. Magnified 215 diameters. Cross nicols.

zinc or mineralogically a calamine instead of M. Gorgeu's willemite which is an anhydrous silicate of the same metal. The water content³⁸ of the crystals, under our chemical examination, did not pass off until heated above the Meeker burner with a strong flame. From these experiments the writer concludes: (1) that M. Gorgeu probably mistook the hydrous silicate for the anhydrous silicate of zinc by neglecting its water content (of which he made no mention in the chemical analysis of his product,

³⁸ According to Fock, "the water goes off only at a red heat, the mineral being unchanged at 340°C. Dana. *System of Mineralogy*, Sixth Edition, p. 548.

and his optical description is rather insufficient); and (2) that he probably took natural calamine instead of willemite (in some instances these two minerals resemble each other and are difficult to distinguish, especially when they are wholly white, and exhibit similar radial structure—as indeed is rare, yet occurs in some of the white willemite from Franklin Furnace, N. J.) for comparison with his synthetic product, since he declared that the elongation of his crystals was parallel to c —which is one of the common characteristics of most of the fragments from the crystals obtained by the writer, and which is very common with natural calamine as well as willemite in thin sections. It, therefore, cannot be used as a criterion for the determination of either mineral.

In one respect this synthetic work on calamine may seem to some of the geologists, to be inadequate, namely the temperature and the boiling-water treatment of formation. Field observation points to ordinary temperature. Doubtless, it would be highly desirable to obtain similar crystals of calamine at ordinary temperatures. It must be remembered, however, as pointed out some time ago by well-known geo-chemists, "that there is nothing in our knowledge of the minerals to indicate anything more than continuous changes in the necessary conditions as the temperature falls." It must also be admitted that to imitate a process in nature will be always more or less a matter of extrapolation.

7b. Possible Effect of Heat on Calamine and Smithsonite.—Experiment 13.—It is known that artificial zincite and willemite occur in furnace slags and in iron and lead furnaces from the ores carrying zinc. It has been suggested by Dr. Kemp that the ores of zinc at Franklin Furnace and Ogdensburg, N. J., are "the result of metamorphism." This suggestion led the writer to investigate the effect of heat on calamine and smithsonite. The experiments described below are of very simple character, consisting of only four steps:

1. Fusion of the Natural Specimens of Calamine and Smithsonite.—Mere heating in a good gas furnace, does not produce any result beyond driving off H_2O from calamine and CO_2 from smithsonite. The resultant products are usually crumbly and brittle, although they still retain the original structures. No optical change was observed. It is worthy to note, however, that when water is driven off from calamine by heating, the product shows the fluorescent property which is commonly used in testing willemite at Franklin Furnace, N. J. The writer obtained the fluorescence by burning "magnesium ribbons" before the sample.

The results of fusion are quite different. The H-O flame was employed for the fusion, and the fused mass was cooled slowly in each case in the neighborhood of the flame. Fragments of the fused portions of the respective minerals were mounted on slides.

2. Examination under the Microscope.—The fused fragments from smithsonite gave uniaxial positive properties indicating a change in op-

tical sign, since for smithsonite, the optical sign is negative. (Out of hundreds of fragments only a few showed this change the rest were overfused and became cloudy under the microscope.) With the fused calamine, some difficulties were experienced, but the writer succeeded in finding some fragments showing uniaxial positive properties, indicating that the mineral had changed its interference figure; since for calamine, the interference figure is biaxial. It was also observed under the microscope in some fragments that the calamine was completely changed being fully composed of minute six-sided crystals. This seems to indicate the molecular change by recrystallization to willemite.

3. Chemical Analysis.—These analyses do not demonstrate that the fused specimens are molecularly changed; they merely indicate the H_2O and CO_2 are absent after the heat treatment.

	Fused Calamine			Fused Smithsonite		
	Per Cent.	Molecular Weight	Molecular Ratio	Molecular Proportion		Per Cent.
SiO_2	27.06	60.4	0.4584	1	ZnO	100.18
ZnO.....	72.93	81.4	0.8960	2	CO_2	None
H_2O	None					

4. Test for Fluorescence.—The fused portion from calamine shows fluorescent color, though less strongly than the willemite from Franklin Furnace, N. J.

These results strongly suggest that under the application of both heat and pressure, as they commonly exercised their influence under metamorphic conditions, the still more striking and definite changes might be exhibited. But this cannot be obtained without the necessary equipment for both pressure and temperature. A system of slow cooling is also essential.

E. Summary of the Experimental Work

The oxidation and solution of blende is prerequisite to the formation of any one of the oxidized ores, and the process of oxidation is brought about by reactions of the sulphide with a ferric iron salt. Ferric sulphate is most conveniently used.

When the iron salt is ferric sulphate the zinc sulphide is usually converted into sulphate; but the solvent action of this salt does not seem to be in the proportion to its concentration, as only a small percentage of the ferric is reduced to ferrous by the oxidation of the zinc sulphide. These results suggest that the chemical equation, $Fe_2(SO_4)_3 + ZnS = FeSO_4 + ZnSO_4 + S$, represents only the chemical reaction and not the chemical ratio.

When a ferrous salt is alone present (as it rarely is, in quantity, in the zone of oxidation), no oxidation of zinc sulphide can take place unless free air is accessible; and when air is introduced, the sulphide may be converted into sulphate by the oxidation products of the ferrous salt prior to the formation of any one of the oxidized ores of zinc.

Soluble salts of such metals as Ag, Cu, and Pb in solution, exert a powerful solvent action on blende. Of these, a solution of silver salt is much more powerful than a copper salt solution and the latter is more so than a salt solution of lead. Lower and different concentrations of the sulphates of silver and copper were used. In case of lead, lead nitrate was substituted.

A solution of cupric sulphate has a powerful solvent action on the natural sulphide of zinc, hence in an orebody containing much copper sulphide or galena or both, little sphalerite may be expected in the horizons where the former two sulphides are rich, as most of the blende will be taken into solution and the metal will be redeposited somewhere as one of the oxidation products; and conversely, if rich zinc sulphide be found in a certain horizon, we may be pretty sure that the orebody cannot have contained much lead (or copper) sulphide, and that little oxidation products should be expected. This accords with the fact that zinc carbonate and zinc sulphide have been found in different horizons, as if they had no relation to each other, as in the Circle Mines at Oronogo, Mo.

When a solution of an alkali or calcium bicarbonate is mixed with a solution of zinc sulphate or chloride, a zinc carbonate is precipitated; but when a reagent containing normal carbonate, such as Na_2CO_3 or K_2CO_3 , is used, only a basic carbonate or hydrozincite is obtained.

When a dilute silicate solution (Na_2SiO_3) is mixed with a dilute solution of zinc salt, only a precipitate of a meta-silicate of zinc is obtained. In the presence of carbon dioxide, this silicate solution replaces also zinc carbonate, to form meta-silicate of zinc. If the reagent contains a mixture of sodium silicate and sodium hydroxide a "complex" precipitate of meta-silicate and hydroxide of zinc may be obtained. In chemical composition this precipitate corresponds to natural calamine, but by reason of its amorphous character no definite confirmation is here offered.

The presence of carbon dioxide exerts a powerful solvent action on all the oxidized ores under consideration. Of these, zinc carbonates are more soluble than the basic silicate of zinc in waters containing an excess of carbon dioxide. This may account for the fact that no carbonic acid, as far as we know, has ever been detected in the analysis of the waters from the mines of these oxidized ores. Upon release of the carbon dioxide any of them may be recovered.

Limestones immersed in dilute solutions of zinc sulphate were found to be replaced by zinc carbonate; but the replacement is much more rapid with the impure limestone than with the pure ones. This result suggests

that the presence of organic matter in the impure limestone has probably exerted some influence in the rate of replacement.

At or a little below $900^{\circ}\text{C}.$, minute crystals of calamine may be prepared from a chemical mixture containing definite proportions of H_4SiO_4 , Na_2SO_4 (dried), and ZnSO_4 (dried). Both the chemical analysis and the optical characters of the crushed fragments all point to their identity with natural calamine.

The influence of heat on calamine and smithsonite results in the change of their optical characters into those of willemite and zincite; but the optical tests were so difficult to carry out satisfactorily that it appears doubtful whether the anhydrous silicate and the oxide of zinc can be formed in this way.

III. PETROGRAPHIC WORK

Introductory

Petrographic data concerning the mode of formation of the ores of zinc are, so far as we are aware, limited to descriptions of occurrences of blende.³⁹ As to the oxidized ores practically no attempt⁴⁰ has heretofore been made to discuss the paragenesis of the mode of their formation in the light of their petrology. They have naturally been regarded as products from the alteration of sphalerite, but the conclusion was wholly drawn from field studies.

The data given below are the results of a study of thin sections from ore specimens, chiefly selected from the collection of ores of the Department of Geology at Columbia University, and representing only a few localities chosen for this purpose.

This was undertaken in conjunction with the experimental work, in order to arrive at some definite conclusion as to (a) the genetic order of the precipitation of zinc carbonate and zinc hydrous silicate; and (b) the influence of inclosing rocks and associated minerals, principally limestone and pyrite respectively, upon sphalerite and its oxidation products. For this reason the descriptions are not necessarily confined to the oxidized ores alone but also include the sulphide; and the specimens from which thin sections were cut are grouped, not according to their localities but according to their present form of occurrence, into two types as follows: (1) Sphalerite occurring in limestones or cherts from southwest Wisconsin; southwest Missouri; Mascot Mine, Tennessee; northwest Arkansas; Wythe County, Virginia; and Saucon Valley, Pennsylvania; and (2) partially altered sphalerite and the oxidation

³⁹ C. E. Siebenthal. *Mineral Resources*, 1911, p. 353 et seq., U. S. Geological Survey, W. Lindgren, *Mineral Deposits*, p. 422 (1913); and T. L. Watson, *Bulletin No. 1; Geological Survey of Va.*, pp. 95 to 96 (1905).

⁴⁰ Watson. *Idem*, p. 45.

products, calamine and smithsonite, from southwest Missouri, northern Arkansas; and Leadville, Col.

1. *The Occurrence of Sphalerite in Limestones or Cherts*

Different views have been advanced by numerous authors as to the mode of deposition of zinc sulphide in limestones and cherts or in their breccias. Later writers have generally agreed, however, that the ores of this type were first derived from above by the decomposition and erosion of superimposed strata, and finally concentrated by meteoric waters in brecciated limestones or cherts.

The result of the present investigation on thin sections from the sulphide ores of zinc points to some such theory. The sections examined show that in many instances the zinc sulphide has filled completely the interstitial space in the brecciated country rocks, principally limestones and cherts. In others, the sulphide appears to have impregnated the country rocks and filled the cavities in the limestones as if it had been deposited with the inclosing rock.

Among the associated minerals are galena, pyrite, magnetite, limonite, secondary quartz, and calcite, fluorite and greenockite. The latter two were not observed in the sections here examined; and galena was absent, or occurred only in minor amount, where sphalerite was dominant. Sphalerite usually incloses galena, indicating the previous deposition of the latter. Of the three iron compounds, pyrite is the commonest, while magnetite infrequently concentrated around the peripheries of sphalerite, and limonite occurs sparingly along cleavage cracks, when such occur. The order of origin of these minerals with respect to the sulphide ore of zinc has therefore appeared to be: (1) Galena (if present); (2) sphalerite; (3) iron sulphides; and (4) secondary quartz and calcite. (2) and (3) may have taken place almost simultaneously and (3) and (4) reverse in some instances. Wherever quartz occurs in these sections, it is invariably of secondary origin.

A specimen from southwest Wisconsin consisted of sphalerite and a siliceous mass on the top, which was thought to be the alteration product, calamine, but upon examination under the microscope was found to consist of aggregates of secondary quartz. Some are cryptocrystalline and others are much larger. In it secondary calcite occasionally occurs (Fig. 6).

The essential relation of secondary quartz to sphalerite appears to be the same in all the specimens studied from the localities indicated above, as shown in Figs. 6 to 11. Irrespective of their localities, they all show that the voids or interstices in the limestone were completely filled with secondary quartz, and that where spaces in the country rock were not wholly occupied by sphalerite the remainder was filled with quartz.

Secondary calcite is occasionally seen in cleavage cracks of sphalerite. These relations indicate that the precipitation and concentration of sphalerite preceded the deposition of secondary quartz and calcite.

Some sections show pyrite penetrating the blende, as well as the calcite and quartz, along their strain- and cleavage-cracks. This means that subsequent to the deposition of most of the sphalerite and of the secondary quartz in the fractured limestone, there came a period of infiltration of solutions, bearing iron sulphide. In these cleavage cracks of sphalerite and quartz, pyrite has been deposited. It further favors the contact of sphalerite and is especially prominent around the edges of the zinc sulphide. The contacts of pyrite and sphalerite thus produced afford the most important factor in the oxidation of sphalerite, discussed in the earlier part of this paper.

Sphalerite is, as a rule, most abundant in brecciated limestone and in the secondary black chert. The latter is largely made up of cryptocrystalline quartz with grains of blende and crystals of dolomite (Fig. 7). The dark color of the chert is due to the presence of much organic matter. In the bedded limestones and white cherts, sphalerite is practically absent.

Summary of the Blende in Limestones and in Cherts.—From the thin sections examined here it may be briefly stated that the deposition of secondary quartz was subsequent to the crystallization of sphalerite. This secondary quartz was probably in part introduced by percolating waters and in part was furnished by the decomposition of cherty limestone and cherts with minor amounts of secondary calcite. The precipitation of pyrite in some cases was almost simultaneous with the crystallization of sphalerite, but it has not had the opportunity since its deposition to exert influence on the oxidation of sphalerite.

2. The Partially Altered Sphalerite and the Oxidation Products

The country rock in which the ores of this second type occur is chiefly dolomitic limestone. The associated minerals are galena, pyrite, limonite, hematite (both derived from pyrite), secondary quartz and calcite, and some primary quartz. In no case are they all present in a single section; and primary quartz and pyrite were observed only in specimens from Joplin district, Missouri, and Silver Hollow Mine, Arkansas.

a. *Smithsonite with Blende* (Fig. 18).—This shows the peculiar occurrence of galena around peripheries of residual grains of sphalerite. In some cases galena forms a succession of rings. The sphalerite within a single ring is usually unaffected. The iron oxide associated with smithsonite is usually in the form of limonite. It penetrated or concentrated along the cleavage cracks of smithsonite, as shown in Fig. 18. A characteristic feature of the smithsonite is the occurrence of minute secondary

quartz and of some straight and elongated crystals of calamine intergrown in it, as shown in Fig. 14. It is evident that the intergrowth or interlocking of these oxidation products (Figs. 13 and 14) represents the overlapping periods of formation of both minerals. Their amount of each necessarily depended upon the amount of the required substances in the reagents in original sections.

The ores from Silver Hollow Mine, Arkansas, contain a large number of almost round grains of quartz, some of which were completely surrounded by pyrite and others by both pyrite and secondary calcite. These quartz grains are primary in origin, and are believed to have come from the disintegration of some original sandstone, before the penetration of pyrite. The edges of these quartz grains are more or less corroded and in some cases secondary calcite has filled these spots (Fig. 15). The optical examination of this ore indicates that smithsonite is rarely derived direct from sphalerite, but mostly from solutions, after the sphalerite had been oxidized into a soluble salt presumably a sulphate, by meteoric waters. Fig. 16, from the same section, shows sphalerite cut by a veinlet of smithsonite. In some places the section also shows the sharp contacts of smithsonite with sphalerite. Had the smithsonite been formed by direct alteration from sphalerite such a relation could hardly exist. These sections further indicate that simultaneous with or subsequent to the solution of sphalerite in some parts of the ore there occurred a fracturing of the unaltered sphalerite, which permitted the infiltration of solutions bearing smithsonite. In the cracks thus produced in the unaltered portions of sphalerite, smithsonite has been deposited. Subsequent to its deposition is the deposition of secondary quartz which filled the rest of the vein. Pyrite which penetrated sphalerite and calcite along their cleavage weakness in the specimen is only observed in part in thin sections.

b. *Calamine with Blende*.—Where blende was not entirely altered, its periphery was surrounded by a ring of galena (see Fig. 18) and occasionally by limonite. Like smithsonite, calamine rarely replaced sphalerite in place. Its crystals usually exhibit a habit of radial or of massive structure. This habit of structure (Figs. 13 and 17) suggests at least one thing, namely, that its crystallization in general took place from solutions, and not by direct alteration of sphalerite.

The rich ore of calamine may also carry some smithsonite as a minor product, and the same is also true in the reverse case. Both of them may replace calcite as the result of metasomatic replacement, as shown on Fig. 18. It is sometimes difficult to distinguish smithsonite from calcite, especially when they are closely associated. The difference in their relief generally affords a clue to their distinction. In nearly all cases, the surface of smithsonite appears very shaggy under the microscope.

c. *Smithsonite and Calamine*.—The sections here examined were cut

from the ores containing both calamine and smithsonite and are practically free from gangue. In some instances they are intergrown in each other; in others they are interlocked with each other, as already described above.

As to their structure, each of these two minerals seems to have followed a definite habit of crystallization. With calamine the radial structure is common, whereas smithsonite is usually granular, especially in high-grade ore (Figs. 13, 14, 17).

A specimen, known to contain both calamine and smithsonite, from Leadville, Col., was found to be mostly calamine, largely made up of crypto-crystalline and elongated crystals, as shown in Fig. 19.

Fig. 20 from the same section shows narrow strips of iron oxides (hematite and limonite) surrounded by calamine. These oxides appear to have concentrated along the partings and the cleavage cracks of calamine. Where this mineral is in the form of minute crystals, the surface is irregularly coated also with iron oxides (Fig. 19), chiefly limonite. The iron oxide was undoubtedly an iron sulphide in origin, and was later reconcentrated and reprecipitated in its present form, after its oxidation product, generally ferric sulphate, had been destroyed in the oxidation of sphalerite, with which it was in contact in the original rock.⁴¹

Summary on the Partially Altered Sphalerite and its Oxidation Products.

—From the examination of these sections it may be said that:

(a) When sphalerite was surrounded by a ring of galena or iron oxide, its further oxidation was terminated.

(b) Sphalerite was rarely replaced by its oxidation products in place. In most cases the solution of sphalerite, through the reaction of the oxidation products of iron sulphides, preceded the transportation or migration, and the deposition of the metal in the form of carbonate or hydrous-silicate.

(c) The alteration of sphalerite proceeded inward zonally from the exterior.

(d) No definite order of the formation of calamine and smithsonite was observed. They may intergrow with each other or either may precede the other. Thus the formation of these oxidation products depended upon the availability of the required substances in the reagents which had come into contact with a solution containing the metal zinc, presumably as zinc sulphate.

(e) Both calamine and smithsonite replaced calcite, or the limestone might pass into these products in orebodies. This relation has also been observed in some zinc mines.⁴²

⁴¹ T. L. Watson. *Bulletin No. 1, Geological Survey of Va.*, p. 45 (1905).

⁴² U. S. Grant. Report on the Lead and Zinc Deposits of Wisconsin, *Bulletin No. 14, Wisconsin Geological and Natural History Survey*, p. 59 (1906).

IV. CONCLUSIONS

From the chemical and petrographic data presented in this paper we have sufficient evidence to justify the following conclusions:

1. Sphalerite is first attacked by the oxidation products of iron sulphides before the formation of the secondary minerals containing zinc.

2. The formation of the oxidized ores rarely occurs by direct replacement of sphalerite.

3. Normal zinc carbonate (smithsonite) is formed when the reagents in solutions are bicarbonates; and basic zinc carbonate (hydrozincite) is formed when the reagents are normal carbonates.

4. Limestones may be replaced in nature by solutions of a zinc salt, usually the sulphate.

5. In nature both smithsonite and calamine replace limestones.

6. No free carbon dioxide can be present where silicates and carbonates of zinc occur, and conversely, these minerals of zinc do not appear where free carbon dioxide is present.

7. "Amorphous calamine" may be formed from solutions of silicate and hydroxide under ordinary conditions.

8. Calamine in nature is probably formed by steps, and is precipitated from waters containing ions of some gelatinous silicate and of hydroxides.

9. It is evident that calamine is formed under ordinary conditions in nature, but time is a great factor in the process. Until this factor is satisfied we can hardly expect the production of crystals of calamine by wet methods.

10. Minute crystals of calamine may be formed by heat at or below 900°C. from a chemical mixture, as set forth under Experiment 12.

11. No confirmation is here offered of the change of calamine and smithsonite to willemite and zincite by heat, other than that heat produces changes of optical characters into those of willemite and zincite respectively.

V. MICROPHOTOGRAPHS AND PHOTOGRAPHS WITH THEIR EXPLANATIONS

Abbreviations

C = calamine

Ca = calcite

D = dolomite

G = galena

I = iron oxides

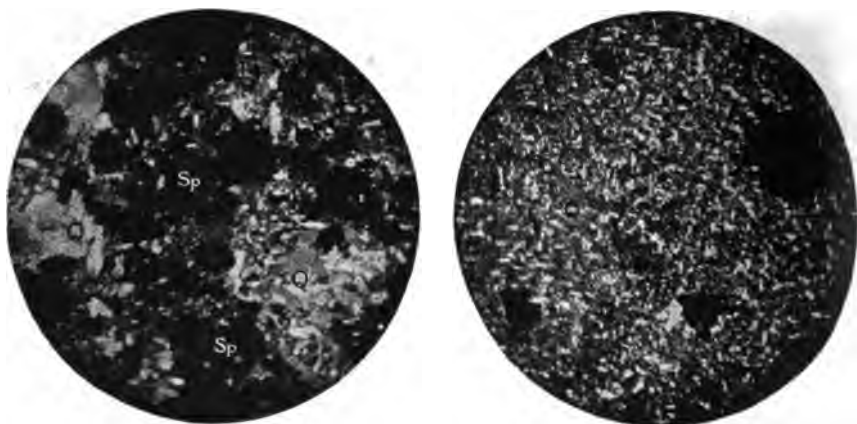
P = pyrite

Q = secondary quartz

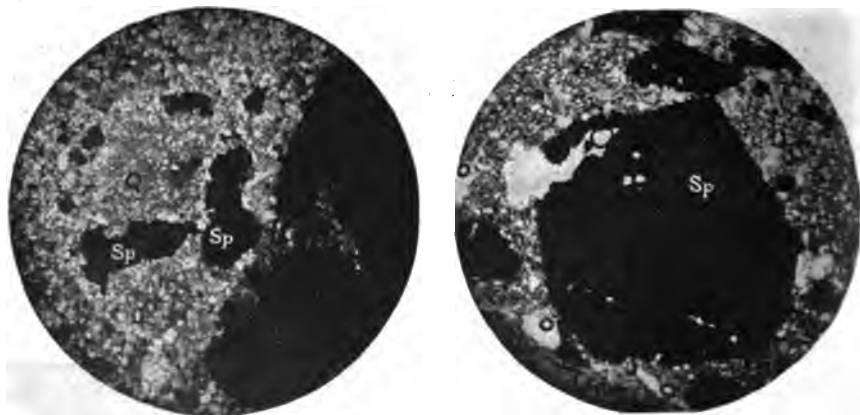
PQ = primary quartz

S = smithsonite

Sp = sphalerite



FIGS. 6 AND 7.—MAGNIFIED 17 DIAMETERS; CROSSED NICOLS.



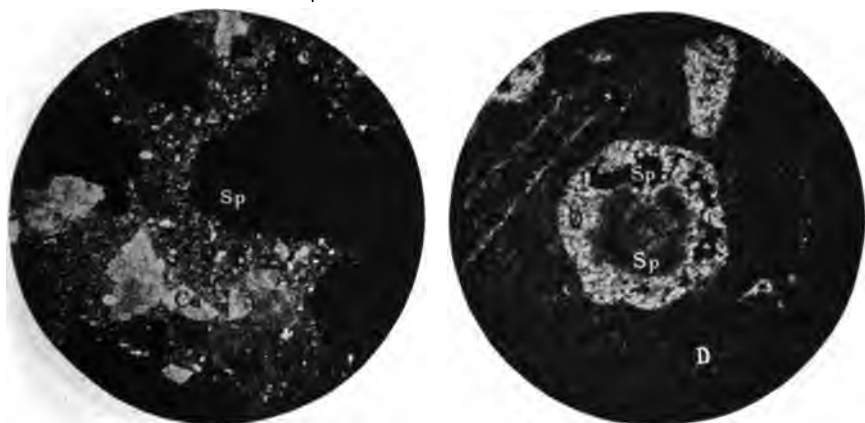
FIGS. 8 AND 9.—MAGNIFIED 17 DIAMETERS; CROSSED NICOLS.

FIG. 6.—Thin sections of blende ore from Southwest Wisconsin. Black areas represent sphalerite. White areas, secondary quartz, which was formerly thought to be zinc silicate such as calamine.

FIG. 7.—Thin sections of black chert containing ore from Orongo Circle Mine, Orongo, Mo. Black areas, sphalerite. Shows cryptocrystalline character of the secondary quartz in the black chert.

FIG. 8.—From the same locality as 7. Shows the voids in the limestone were entirely filled with both sphalerite and secondary quartz.

FIG. 9.—Thin sections of the sulphide ore from Joplin, Mo. Note the sphalerite was entirely surrounded by secondary quartz.



FIGS. 10 AND 11.—MAGNIFIED 17 DIAMETERS; CROSSED NICOLS.

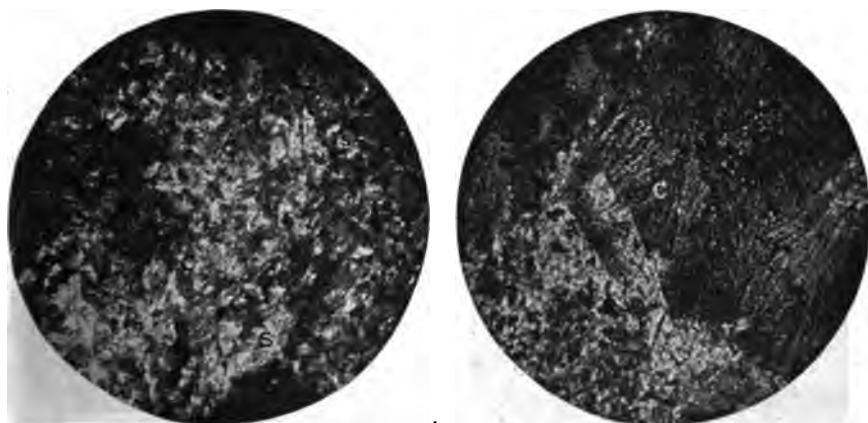


FIG. 12.—MAGNIFIED 17 DIAMETERS;
CROSSED NICOLS.

FIG. 13.—MAGNIFIED 140 DIAMETERS;
PLAIN LIGHT.

FIG. 10.—Thin section of the ores from Northwest Arkansas. Shows the secondary quartz and the filling of minute cracks accompanied by replacement of the sulphide by the quartz. Re-entrant angles along some margins of the sulphide are well shown.

FIG. 11.—Thin section of the ores from Mascot Mine, Tenn. The dark area within the lighter circular portion represents sphalerite. A large portion of it was probably removed by a siliceous solution from which the aggregates of secondary quartz deposited around the remnants of the sulphide.

FIG. 12.—Thin sections of the oxidized ores from San Francisco Mine, Missouri. Shows the intergrowth of calamine and smithsonite. The latter constitutes the major portion of the section including the dark areas; while the latter is only in minor amount and mostly in elongated form of minute crystals.

FIG. 13.—From the same locality as 12. Shows the growth of smithsonite in calamine. Note the difference in their structure. Calamine exhibits its radial habit of crystallization, whereas smithsonite is in lath shape.



FIG. 14.—MAGNIFIED 17 DIAMETERS;
PLAIN LIGHT.

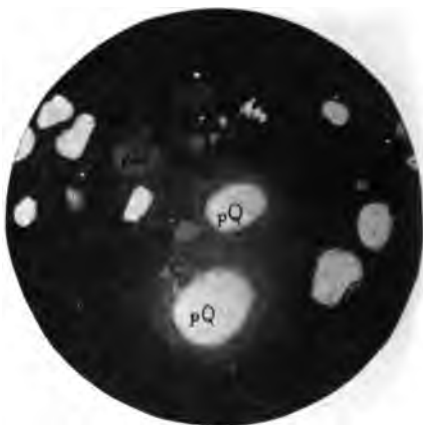
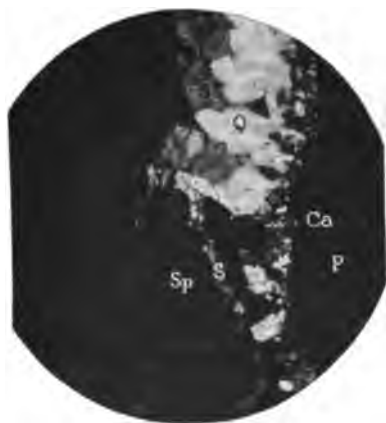


FIG. 15.—MAGNIFIED 17 DIAMETERS;
CROSSED NICOLS.



FIGS. 16 AND 17.—MAGNIFIED 17 DIAMETERS; CROSSED NICOLS.

FIG. 14.—From the same locality as 13. Note the interlocking of calamine and smithsonite. The darker surface of calamine is due to the cloudy appearance of the crystals.

FIG. 15.—Section of the ore containing blende and smithsonite from Silver Hollow mine, Arkansas. Note the round grains of quartz and their corroded edges filled with pyrite and secondary calcite.

FIG. 16.—From the same locality as 15. Note the veinlet of smithsonite in sphalerite and their sharp contacts. In this, the oxidized ores had probably removed only a short distance and almost remained in places, but it nevertheless indicates the fact that the solution of sphalerite preceded the deposition of smithsonite.

FIG. 17.—Thin sections of calamine ore from Joplin, Mo. Note the radial habit of growth in almost all directions.



FIG. 18.—MAGNIFIED 17 DIAMETERS;
PLAIN LIGHT.



FIG. 19.—MAGNIFIED 17 DIAMETERS;
CROSSED NICOLS.

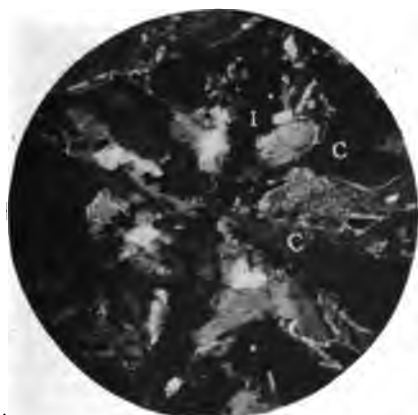


FIG. 20.—MAGNIFIED 17 DIAMETERS; CROSSED NICOLS.

FIG. 18.—Thin sections of the ore specimen from Northwest Arkansas. Shows the galena "rings" in dark color inclosing the residual grains of sphalerite. The areas outside the innermost "rings" were replaced by smithsonite. Note also the replacement of calcite by smithsonite to the right. This probably indicates the fact that upon the solution of sphalerite, a carbonate solution immediately came in and gave rise to the precipitation of zinc carbonate. See discussions under the formation of zinc carbonate in the earlier part of this paper.

FIG. 19.—Thin section of the oxidized ores from Leadville, Col. Note the two types of crystals; one is in an elongated form and the other is mostly cryptocrystalline. Its cryptocrystalline character almost resembles that of some of the secondary quartz in the blende-bearing cherts. (See Microphotograph No. 7)

FIG. 20.—From the same locality as No. 19. Note the narrow strips of iron oxides concentrated along the crystallographic partings and the cleavage cracks of calamine.

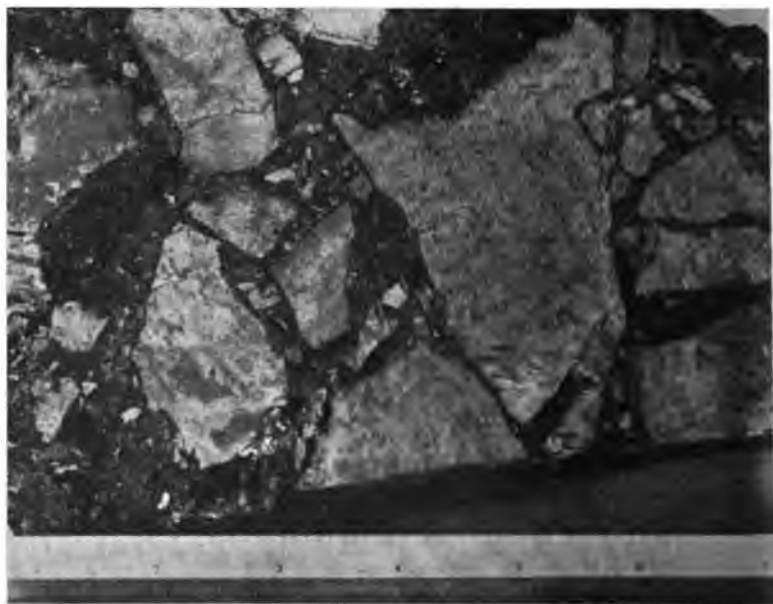


FIG. 21.—Specimen of blende ore from Oronogo Circle Mine, Oronogo, Mo. Brecciated structure of the country rock. Dark areas represent sphalerite and the fine grains of black chert cemented together. The lighter portions represent brecciated limestones and white cherts.

FIG. 22.—From the same locality as No. 21. The white areas represent white cherts in which practically no sphalerite occurs.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, Sept., 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

The Stresses in the Mine Roof

BY E. DAWSON HALL,* BROOKLYN, N. Y.

(San Francisco Meeting, September, 1915)

THE stresses in the simplest structures are often those we find most difficult to analyze. The most complex condition in mine stresses is found in simple tunnels where the roof, the sides, and the floor are a monolith. The functions of the parts are like the parts themselves not distinct and specialized, and the problems to be solved are like those in a metal structure with riveted joints or a redundancy of bars.

This difficulty explains perhaps why the condition has not been treated. But just because it cannot be discussed in its entirety is no reason why it should be treated as an action of parts with specialized functions as a roof beam with supports and a foundation. The problem cannot be ignored on the ground that it is not of sufficient importance to warrant careful consideration, because conditions of complete monolithism, of which the tunnel is the type, are found materially unchanged in room-and-pillar and in longwall work.

This unity between roof, sides, and floor, which to the coal miner is a difficult conception, really deserves a scientific appellation, and perhaps holoid (from *holos*, whole, and *eidos*, form) will serve the purpose as well as any other.

In a simple tunnel the roof, the sides, and the floor form integral parts of one and the same structure, and the distortion of one cannot be conceived without a consequent strain in the others. Thus when the roof of the tunnel droops by reason of its weight, the upper parts of the sides are drawn in because they are integrally connected with the roof and must approach each other whenever, by the sagging of the roof, the distance between any two points in it is diminished. (See Fig. 1.)

The sides in their turn operate on the floor of the holoid structure, producing a tensile stress. The writer has always been impressed with the value of soap as a means of illustrating the action of mine stresses. With that idea in mind a cake of naphtha soap measuring $4\frac{3}{8}$ by $2\frac{1}{4}$ by

*Associate Editor, *Coal Age*.

1 $\frac{3}{4}$ in. was taken and a tunnel was made through it 1 $\frac{3}{4}$ in. long, 2 $\frac{1}{2}$ in. wide and 1 $\frac{1}{4}$ in. high. (See Fig. 2.) A load was then placed at the mid-span of the tunnel. Eventually, the upper bar, or "roof," broke at the center line and along both "ribs" of the tunnel, the breaks being approximately vertical and proceeding, as might be expected, on the ribs from the "surface" downward and at the center line from the tunnel upward, the failure being from bending moment, not shear.

This is interesting because it shows that breakage *at the ribs* is not necessarily evidence of shear. It may be only a demonstration of a holoid structure. It is the form of failure whenever coal is blasted down and not an infrequent form of roof demolition. The test on the soap tunnel further showed that as soon as rupture takes place in the roof of the tunnel, there must inevitably come a thrust on the ribs. The tension draws them together till rupture occurs and then the two roof units, in endeavoring to revolve, crowd each other and push on the opposing walls. (See Fig. 3.)

When a holoid structure, by reason of the weakness of the floor or because of a lack of adhesion between the ribs and the floor, ceases to engage the floor in its movements, then its shape as a structural element roughly resembles the Greek letter π and for want of a better name we might term the new element a pyoid structure. (See Fig. 4.) The doctors use the word for a totally different purpose with little excuse. "Pyonoid" is the word which they should use to express the attributive "pus-like."

By reversing the soap tunnel after the roof has been caved the weakness of the pyoid structure is made clear. As soon as pressure is brought on the new roof (formerly the floor of the tunnel), the two ribs are seen to recede markedly and if the floor were intact this would result in a well-developed stress in that element of the holoid structure. (See Figs. 5 and 6, showing progressive demolition.)

Probably it is well here to express a belief in the importance of the holoid. The general notion is that all the beds shear horizontally along the lines of stratification and that it is a mistake to consider the mine or even the roof as a monolith. It is true that most of the accidents in mines are due to the lack of unity in the roof. What we call draw-slate accidents are almost wholly due to this fact. Nevertheless, it is interesting to note how firmly roof and coal are usually "burned" to one another. Even when undermined and sheared on both sides the coal often fails to fall, being supported by the vertical shearing strength of the one side still attached and by the adhesion to the roof. The writer is hardly prepared to state when the holoid structure ceases to exist, and of course the time and conditions will vary with the materials under consideration.

It is obvious that with the holoid structure, the ribs being drawn

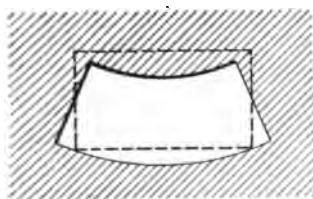


FIG. 1

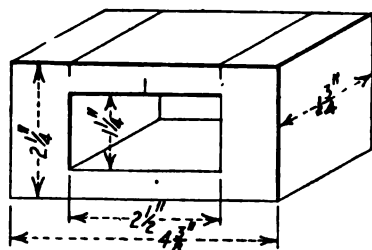


FIG. 2

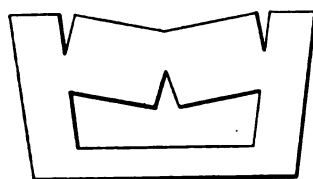


FIG. 3

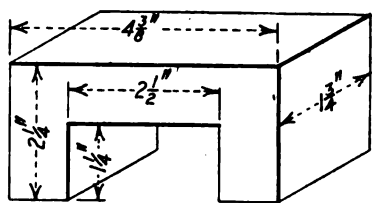


FIG. 4



FIG. 5

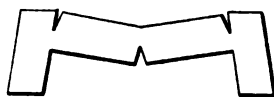


FIG. 6

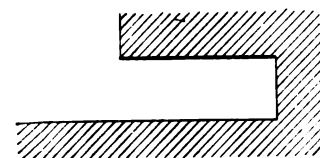


FIG. 7

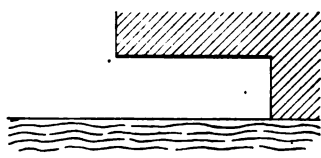


FIG. 8

Fig. 1 shows in broken lines a cross-section of a tunnel and in full lines the same tunnel distorted by pressure. This is a typical holoid structure.

Fig. 2 shows a soap model to which reference is made in the article.

Fig. 3 exaggerates the upper or roof bar in the soap model and shows how the collapse of the roof forces back the sides by the arch action.

Fig. 4 shows the soap pyoid without load.

Fig. 5 shows the same when the bending of the roof bar takes place, thrusting out the ribs, or legs.

Fig. 6 shows a further action when the revolution of the segments in collapse tends to force back the upper parts of the ribs or legs.

Fig. 7 illustrates a semi-holoid structure.

Fig. 8 illustrates a semi-pyoid structure.

together by the movement of the roof, they must tend more or less to be split vertically and in longwall they will then fall down on the advancing undercut. In certain sub-bituminous mines the writer has noted a tendency toward what he thought was a vertical shear parallel to the headings. This developed rapidly after the work was opened, especially at great depth.

Whether this was only a vertical shear seems doubtful. It may have been due to the holoid character of the structure, the ribs receiving no relief by a horizontal shear between ribs and roof. Instead the roof pulled off the edge of the pillar as the former bent under the load. It was interesting to note that these lines of fracture did not coincide with the normal cleavage of the coal.

This rending along vertical planes eventually throws back the real rib lines far into the pillar. Where the draw slate and roof proper leave one another, we have probably a plate structure superposing one which is holoid or pyoid in character, and in the longwall the breaks back of the face which bring coal and roof down together, or which tend so to do if the latter is not duly propped, are failures of the semi-holoid or semi-pyoid and not of the plate structure above. (See Figs. 9 and 10.)

The breaking of even the holoid roof is not necessarily a sudden, unheralded event, such as one might anticipate from a cursory consideration of the problem. It is clear that the action of the moments cannot destroy the roof without revolving in a degree the elements into which the roof is broken, and any revolution inevitably binds these elements against one another so that they are less able to fall. Either a recession of the ribs or a further demolition of the revolving elements must take place or the roof will not fall. One form of demolition which frequently occurs in shallow workings is vertical shear along the cracks already made by the bending-moment stresses. But horizontal shears may make it possible for the roof masses to revolve and yet fit the space they occupied by a counter revolution of the strata past each other. Or again, the whole mass may be broken up by the rubbing of the opposing faces of the elements as they try to fall.

It is this last action which is in evidence in coal brought down by a shot when it is broken considerably in falling, and that vertical shear is not an important cause of the fall of coal is shown by the fact that there is a distinct tendency for the coal to roll away from the side ribs.

It is necessary now to consider the plate structure in which the roof is considered as a vast plate, a monolith in itself but resting without adhesion on its supports. Whenever there is a mined area the roof is depressed, and being elastic it tends to rise on the surrounding supports, resting its weight on the edges of the surrounding ribs of the excavation. This area of quasi-elevation is followed by another area of depression

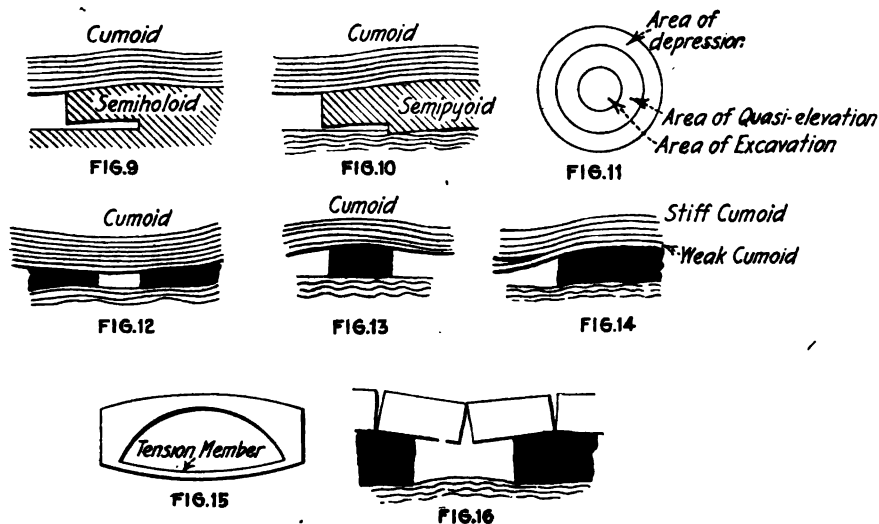


Fig. 9 shows a semi-holoid such as often occurs in longwall, supporting a cumoid which, being separated from the semi-holoid, has freedom of movement except for the restraint of frictional contact.

Fig. 10 shows a semi-pyoid likewise surcharged by an independent cumoid.

Fig. 11 is a plan of a circular excavation. The sagging of the roof into this area produces an area of quasi-elevation; that is, an area which would be elevated if the mineral itself and the floor surrounding the excavation were incompressible and indestructible. Outside this area is one of depression, the stiffness of the measures lightening the burden on the area of quasi-elevation and loading with increased heaviness the edges of the excavation and the area of depression. It exhibits the fact that the action of the roof is cumoidal, or wavelike, and not confined to the area of excavation. The depressed area is in its turn followed by another area of quasi-elevation, and this again by an area of depression. The size of these waves of course decreases as the area of excavation is left behind.

Fig. 12 shows a cumoid bending over an opening and crushing two pillars.

Fig. 13 shows a cumoid bending over a pillar, the center of which is a point of quasi-elevation. The roof tends to break over such a support.

Fig. 14 shows a weak cumoid, such as a loose draw slate which has broken away from the stiffer cumoid above and probably has insufficient moment of inertia for self support.

Fig. 15 is a Luten bridge, showing the lower tension member, or paving, which ties the abutments together and prevents their recession, thus adding to the strength of the bridge.

Fig. 16 shows a broken cumoid failing to fall owing to an arching action.

surrounding the central depression and the area of quasi-elevation. Thus the roof plate is bent into a series of waves around the central area of disturbance just as the surface of the water is rippled round the point where a stone has fallen and disturbed its equilibrium. (See Fig. 11.)

Of course, the elevations are only relative, not actual, and naturally, like all undulations, as they recede from the point of disturbance they die down. But it is essential to remember that while the holoid structure is to a large extent a closed force chain, this is not nearly so true of the plate, the stresses in which are less localized and circumscribed.

This structure we may dub as cumoid (from *kuma*, a wave). The remarkable feature about such a structure is that it develops points of great stress far away from the disturbing cause, and it may break over the pillar instead of in the opening. The appearance of the Forth Bridge, Scotland, is known to almost every one. The light structural work over the midspan contrasts most forcibly with the heavy structure over the piers. It is the case with all cantilevers and continuous-arch structures, and the roof in the mine is like a continuous arch, only it is continuous not only in one, but in every direction.

It is the peculiarity of the cumoid structure that the stresses it involves may be greater farther from the point of disturbance than at some nearer point. But, like the holoid structure and the pyoid, it puts the greater burden on the pillar's edge. The center of the pillar between two large open spaces may be relieved from much of the normal pressure because of the bending of the cumoid roof over the pillar. (See Fig. 13.)

There is some evidence that in actual operations in some sections of the country the roof soon breaks by horizontal shear into two or more separate cumoid structures, of which of course one is free of external load, while the others, though below other cumoids, may or may not be loaded. If the stiffness of the upper cumoid, or cumoids, exceeds that of the lower, the loading may be relieved from the open spaces and the upper cumoids may restrain the lateral, and therefore the vertical, movement of the lower cumoids, thus adding to their resistive strength.

In an interesting paper read before the winter meeting of the West Virginia Coal Mining Institute, the late F. C. Keighley called attention to the fact that when the lower roof broke or was preparing to break in the mined spaces of the Connellsville region it frequently weakened the lower roof in the rooms and headings nearby, despite the strong support afforded by large pillars. This caused in the narrow places many falls which had to be loaded out.

It would seem, therefore, that the breaking of the lower roof or its initial stressing tears the lower roof from the upper and from the ribs, converting it into a cumoid structure which is too weak to stand the strains to which it is exposed. In cases, however, these primary failures

may be due to substitution of a pyoid for a holoid structure. For it must not be forgotten that where the bottom, ribs, and roof are one and indivisible, the floor is an element of strength and prevents the roof from breaking. Just as the lower flange in a rail helps the ball of the rail to support the weight, so does the floor help sustain the roof so long as the former is unbroken.

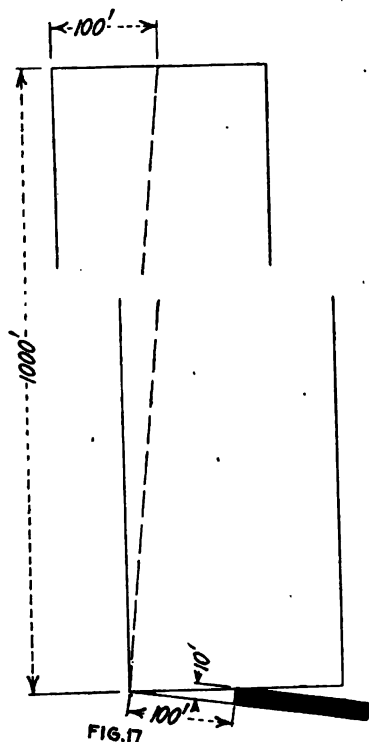


FIG. 17

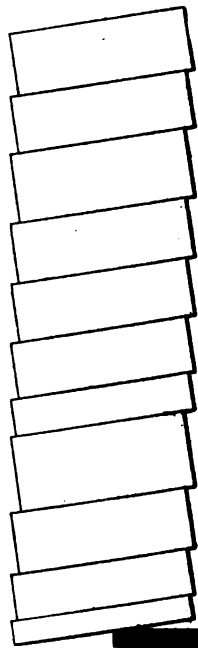


FIG. 18

Fig. 17 shows the large amount of distortion needed to permit the fall of one leaf of a broken cumoid, or conchoid, 1,000 ft. deep.

Fig. 18 shows how this distortion is secured by repeated horizontal shears at places of weakness.

One of the patented features of the Luten bridge construction is its holoid structure. Luten does not use the term, but nevertheless the construction is closely analogous to the structure I have been describing. Luten builds not one bridge but two, one for vehicles to pass on and one to bind the feet of the piers together so as to form a perfect holoid. Destroy the lower bridge and the upper or traveling bridge is in a precarious condition. (See Fig. 15.)

The holoid structure gives the strongest of roofs; the pyoid is less strong; the cumoid is still weaker. Every horizontal shear makes the roof progressively less stable. Strange to say, vertical cracks permit the replacement in many cases of a less stable by a more stable structure. The fractures in a vertical direction, which would ordinarily be thought more detrimental than those in any other direction, prove not unimportant, it is true, yet in a way strengthening.

When a cumoid of great depth finds itself inadequate to its own support as a cumoid and begins to rend vertically from the surface downward to its supports and upward from the span centers to the surface, then the cumoid beam becomes an arch, and the cumoid plate becomes a dome and until it fails in its new structural relations, it cannot continue to extend those rents which threatened its stability. Let us say that the cumoid has become a conchoid (from *conchus*, a shell), the lines of stress forming figures resembling the shell of a bivalve.

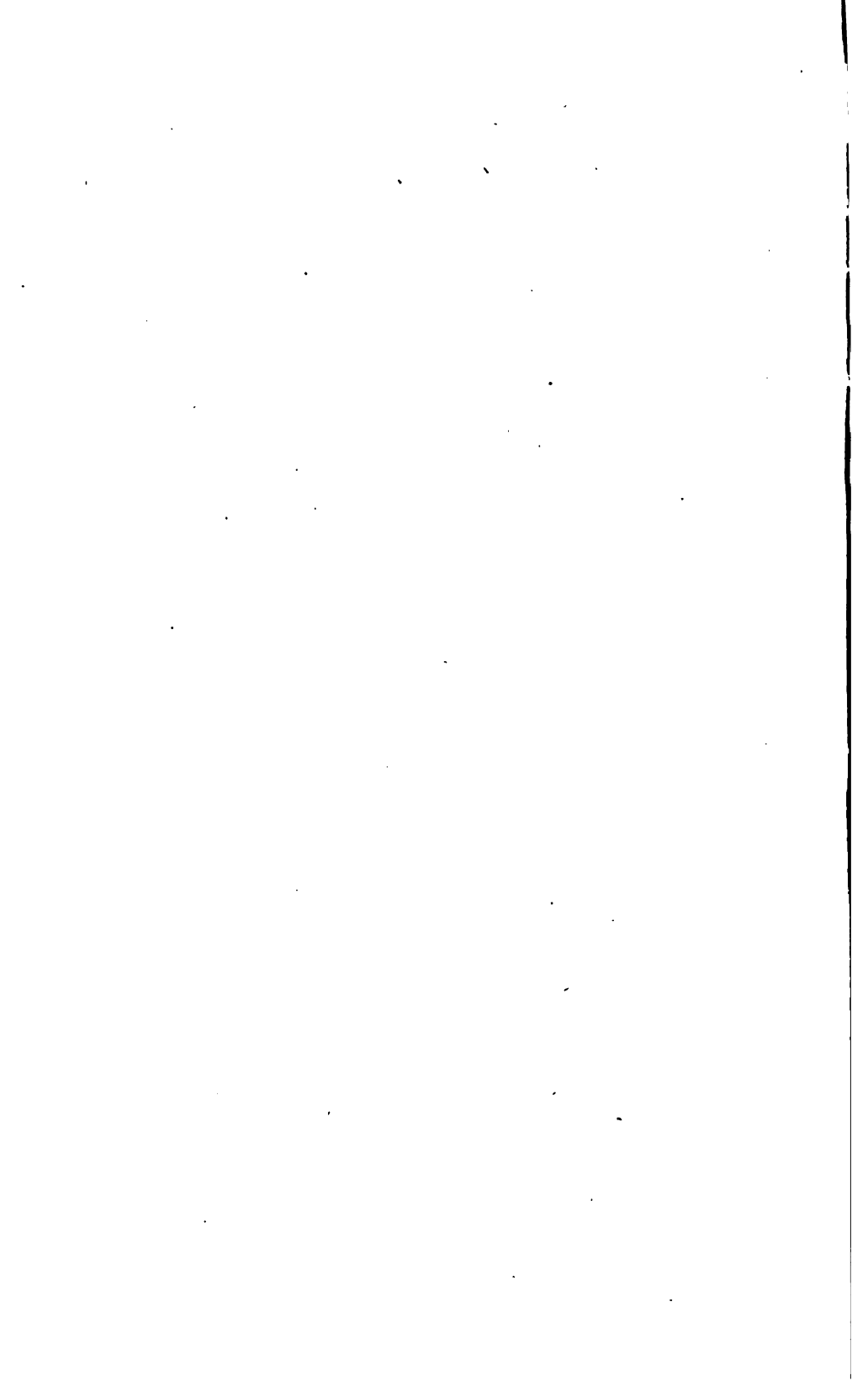
As the elements of which the roof consists crowd each other into the opening it is impossible for the roof to rupture or fall until the various strata cease to act as a unit and begin to slide past each other. (See Fig. 16.) If we imagine an element of rock stretching from a rib to the half span, a distance, let us say, of 100 ft.; if we further suppose the excavated coal or other mineral to be 10 ft. thick, and regard the rib, floor, and falling rock element to be so adamant that none of the three will crush, heave or bend, then the inclination of the element when one corner reaches to the floor will be 10 per cent.

If the excavation is 1,000 ft. below the surface, the displacement of the block at the surface would have to be 100 ft. (See Fig. 17.) At a greater depth the displacement would be more. It is needless to say such a displacement cannot occur even where the crop, broken roof, or the sags in nearby workings make a certain amount of lateral adjustment possible. But horizontal shear along weak planes, such as beds of clay or coal or stratification planes, will permit such a number of readjustments in the element of rock itself that it will be able to come down. It will, in fact, rather *tend* to tilt than actually upset forward. (See Fig. 18.)

This, then, is how the end comes unless the rock is too strong to shear horizontally. Subsidence ceases when the required deformation has taken place. The choking up of the falls probably has but little to do with the final result, though it may have its effect in some cases. In fact, the theory as given is comforting to the conservationist, as the beds above the one extracted suffer but little, being exposed only to the horizontal shearing process, which is by no means destructive of the integrity of the measures.

However, the rupture at the half span may often result in falls of moderate size at that point, for when the monolithic roof is weakening by

successive horizontal shears, we have a series of superposed cantilever cumoids entirely unequal to the task of self support. If the action which forms these cumoids does not let them all down to the floor suddenly, they will be sure to fracture successively from the bottom upward, and such seams of mineral as partake of this independent action will be destroyed.



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Kick vs. Rittinger: An Experimental Investigation in Rock Crushing, Performed at Purdue University

BY ARTHUR O. GATES,* B.S., M.E., MISHAWAKA, IND.

(San Francisco Meeting, September, 1915)

INTRODUCTION

RITTINGER'S law of the energy expended in crushing is, as roughly stated by Professor Richards,¹ that the work of crushing is proportional to the reduction in diameter; or, as I have more fully expressed it:

"The work done in crushing is proportional to the surface exposed by the operation; or, better expressed for this purpose, the work done on a given mass of rock is proportional to the reciprocal of the diameter of the final product—assuming that all the mass has been reduced to one exact size, which is only theoretically possible."²

Kick's law is stated by H. Stadler as follows:³

"The energy required for producing analogous changes of configuration of geometrically similar bodies of equal technological state varies as the volumes or weights of these bodies."

In other words, the energy expended is proportional to the volume reduction, instead of the diameter reduction.

That these two laws would give widely different results may be shown by a simple imaginary case. A ton of 16-in. cubes is broken to 1-in. cubes at the first operation, and these are broken to $\frac{1}{16}$ -in. cubes at a second operation. Since the first operation produces only one-sixteenth as much new surface as the second, the ratio of energy expended in the two operations would be by Rittinger's law as 1 to 16. By Kick's law, since the volume-reduction ratio is the same in both cases, the ratio of energy would be 1 to 1. This discrepancy is great enough to challenge a test by actual experiment, of which there has been, hitherto, but little compared with the amount of argument. The present paper is the record

* Sales Engineer, Dodge Sales & Engineering Co.

¹ *Ore Dressing*, p. 304 (1903).

² *Engineering and Mining Journal*, vol. xciv, No. 21, p. 1039 (May 24, 1913).

³ *Grading Analyses and Their Application, Transactions of the Institution of Mining and Metallurgy*, vol. xix, p. 478 (1910-11). The original statement will be found in *Das Gesetz der proportionalen Widerstände*, by F. Kick.

of such a test, and is offered as showing that Rittinger's theory more nearly represents the actual facts than any other proposed hitherto.

Mr. Stadler, in the paper already cited, applying to rock fracture Kick's law (which I accept as applicable to the deformation of elastic bodies), proves that the stamp mill, working on relatively coarse feed, is several times as efficient as the tube mill on fine feed, this relation being expressed in what he calls "relative mechanical energy per horsepower." He distributes his energy units among the various sizes produced, according to Kick's law. But my results show, among other things, that many more of Stadler's energy units are obtained per foot-pound-applied in coarse than in fine crushing. This suggests a doubt as to the correctness of his unit, since one would expect a foot-pound-applied to produce the same number of "energy units," whether the feed and resulting product were coarse or fine. When Rittinger's theory is applied, the number of mesh-tons (my unit of surface produced) is nearly proportional to the foot-pounds-applied, whether the product be coarse or fine; hence my conclusions favor the simple Rittinger theory that surface produced is proportional to energy applied.

THE TWO LAWS

1. It is the purpose of this paper to record and interpret a series of experiments made for the purpose of studying the consumption of energy in the crushing or fracturing of rock particles both coarse and fine. At the start, the expectation was to determine constants according to the Rittinger theory for several rocks, whereby it would be possible to calculate absolute crushing efficiencies of machines crushing such rocks, or to predetermine what a machine would accomplish in the way of tonnage and screen analysis upon a given rock, knowing what the machine accomplished upon another rock whose constants were likewise known. While approximate constants for a few rocks were determined, the results indicated that an amount of work (perhaps with the microscope upon sizes too small to be screened) greater than could be accomplished by a single investigator would be necessary to determine these constants with reasonable accuracy. For this reason no constants are submitted as such in this paper.

2. It is possible, however, to compare by means of these experiments the two antagonistic "laws" of crushing already mentioned. The fracture of cubes by compression in a testing machine produces, in nearly all cases, pyramids of the same general shape (Fig. 1), indicating that the stresses are greatest along certain planes of fracture; and, since the relative movement must have been greatest along these fracture or shear planes, the greatest amount of energy must have been expended there. While I know of no tests that show that the several pieces return to the

dimensions existing before the break, it seems reasonable that they do, and that they do not retain the deformation. All that has resulted from the pressure and consequent movement is *cracks*. They are new, while the particles resulting are of the same form and specific gravity as they were before separation, and if put together accurately the dimensions of the mass would be the same.

Rittinger's Law

iii.

3. Rittinger's theory is the law of the *cracks*. It is well explained in an article by E. A. Hersam in the *Mining and Scientific Press*, an abstract of which is given in the third volume of Richard's *Ore Dressing*, and to which I acknowledge my indebtedness for some of my ideas and methods.

4. The mathematics of this theory follow:

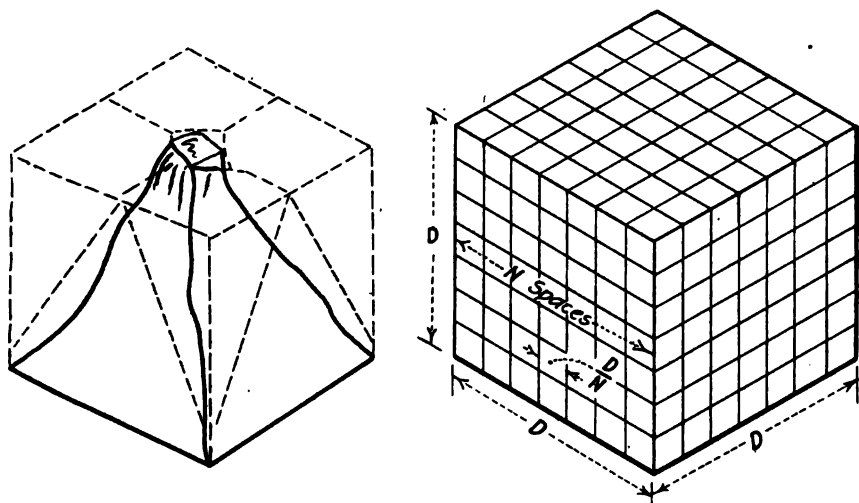


FIG. 1.—FRACTURE OF CUBE IN PRACTICE. FIG. 2.—THEORETICAL FRACTURE.

Assume a cube of rock (Fig. 2) of side D , divided into smaller cubes by planes passing through it in three directions, as shown. The surface of the original cube being $6D^2$, the aggregate surface of the smaller cubes will be $6ND^2$, and the new surface exposed will be $6(N-1)D^2$. The theoretical mesh or reciprocal of diameter of new particles, M , is $\frac{N}{D}$ whence $N = MD$; and the aggregate final surface, $S = 6ND^2 = 6MD^3$.

With D constant, that is, starting from the same size of cubes in both cases, $S : S' :: M : M'$, or the total surfaces of equal volumes of rock, each composed of particles of uniform size and shape, are to each other as the theoretical mesh or the reciprocal of the diameter of the particles composing the respective masses.

Suppose the surface S to be made up of x_1 parts (by weight) of M_1

mesh (theoretical), x_2 of M_2 , x_3 of M_3 , etc. x_n of M_n . Then,

$$S = k \sum_i x_i M_i;$$

that is, the summation of the weights by the theoretical meshes, where k is a constant, six times the volume of unit weight of the rock. If different rocks were considered, specific gravity would enter into the value of k .

Likewise,

$$S' = k \sum_i x'_i M'_i$$

Now where S was the surface existing on the mass before breaking, and where S' is the total surface existing after the breaking, the new surface

$$S' - S = k \sum_i (x'_i M'_i - x_i M_i)$$

For a given screen analysis, weights of the different theoretical meshes, we may plot on co-ordinate paper x against M and x' against M' , and obtain a crushing-surface diagram (Fig. 3) representing the equation

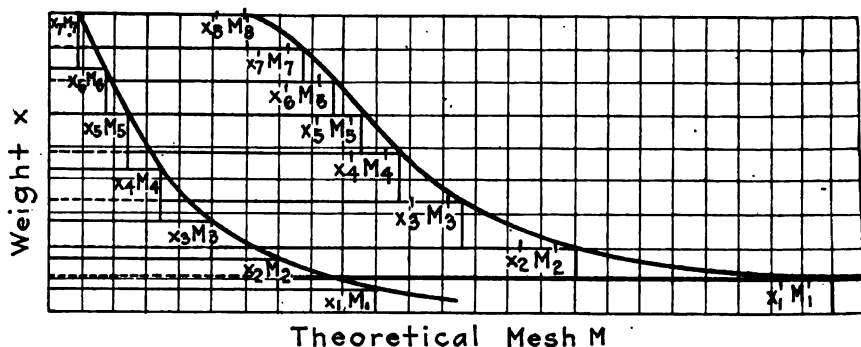


FIG. 3.—CRUSHING-SURFACE DIAGRAM.

above given for new surface, the area between the steps being proportional to the difference in surface area. If we take closer divisions of M , smaller differences in x will result, and the limits of such divisions will result in the smooth curves, the area between which is a true measure of the surface produced.

Thus, on the Rittinger theory that surface produced is proportional to energy applied, the area of the diagram is a measure of the energy going into the crushing. (It should be distinctly understood that bearing- and gearing-friction losses should not be included in the energy-applied measured by the crushing-surface diagram, and that it is therefore impossible to compare the two laws by tests upon commercial machines, unless the friction losses are known.)

THE CRUSHING-SURFACE DIAGRAM

The crushing-surface diagram has the advantage over any other graphic method of comparing screen analyses, that it shows at a glance

the "classification" of the pulp, the distribution of the surface, and (according to the Rittinger theory) the energy spent in its production. As the cards of a steam engine enable us to determine the power developed, and to locate faulty operation or design, so the crushing-surface diagram enables us to determine the power consumed, and to locate points at which the crushing practice may be improved and faults remedied in design.⁴

5. From my original article on The Crushing-Surface Diagram,⁵ I take Fig. 4 and the following theoretical proof of the Rittinger theory:

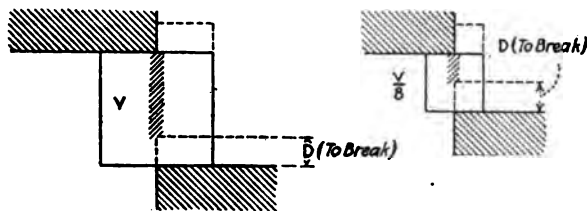


FIG. 4.—RITTINGER'S LAW.

100-Mesh Cube

Area one section = A

Average resistance to shear per sq. in. = F

Energy = FAD

Surface produced = $2A$

200-Mesh Cube

Area one section = $\frac{A}{4}$

Energy = $F\frac{A}{4}D$

Surface produced = $2\frac{A}{4}$

∴ Energy Proportional to Surface.

Suppose it were possible to hold two similar cubes between the two offset faces as shown, and that forces were applied until the deformation shown by the dotted lines was obtained, it will be seen that only the molecules along the vertical center line [it should read plane] are stressed and deformed, the mass of the cube away from this surface receiving practically no pressure or deformation. The energy in this case required to produce rupture will be the product of the average resistance to shearing per square inch by the area along which rupture takes place, and by the distance the two offset faces move together. The average resistance to shearing is a variable quantity, as the deformation increases up to rupture. To reduce to cubes, this amount of energy must be multiplied by three, as three similar fracture planes must be made to produce cubes. And it will not be hard to see that the distance through which the offset faces must move in either case must be the same and not proportional to the thickness of the piece. To break the molecular bond between adjacent particles would require the same movement, regardless of the thickness of the piece.

"It will be noted that if eight of the half-diameter cubes be sheared to produce cubes of half their size, the new surface presented will be double that formed when the single large cube is sheared into half-size cubes, and also that the energy required in the case of the smaller cubes is double that required in the case of the larger cubes. This should demonstrate that energy applied to crushing is proportional to the surface produced."

⁴ This matter has been discussed in my article, Applications of the Crushing-Surface Diagram, *Engineering and Mining Journal*, Apr. 18, 1914.

⁵ *Eng. and Min. Jour.*, May 24, 1913.

Of course the above passage states a theoretical case of shear, and it is not to be expected that cubes can be held in the manner shown; in practice the breaks occur, perhaps, as in Figs. 1 and 5.

6. New units are apparently necessary when using the crushing-surface diagram. The term "mesh-grams" as a proposed unit for crushing constants and measurement of surface I have defined as "the surface produced by a theoretical crushing operation in which one gram of particles of the same diameter are all reduced to a diameter whose reciprocal is one greater than before reduction." The unit "mesh-ton" I have similarly defined as "the increased surface produced by crushing

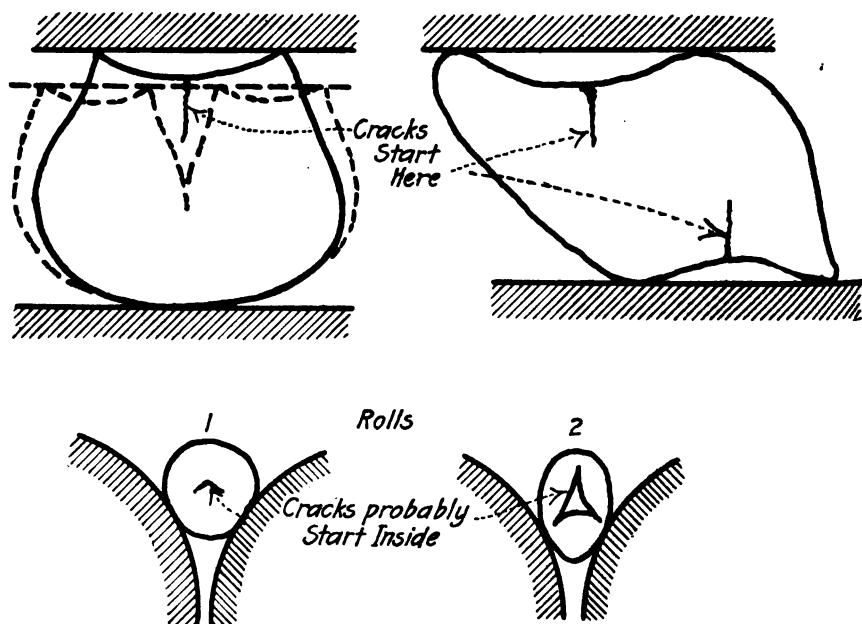


FIG. 5.—BEGINNINGS OF FRACTURE.

all particles of a ton of rock to a diameter whose reciprocal is one greater than in its previous condition.”⁶

As the crushing contemplated in the definitions is only theoretically possible, a couple of examples may explain them better. Shapes remaining the same, preferably cubes for argument's sake, during all stages of the crushing, if 1 g. of 0.1-in. rock (reciprocal, 10) be crushed until it is all 0.05-in. (reciprocal, 20), the new surface produced is 10 mesh-grams. Or if 10 g. be reduced to exactly 0.01-in. from exactly 0.1-in. particles, there will be an increase of 10 times 100 minus 10 times 10, or 900 mesh-grams. Mesh-pounds, mesh-ounces, mesh-percentages, etc., are similarly justified and defined.

⁶ *Trans.*, xlvii, 59 (1913).

7. The term "mesh" for a measurement has been objected to, as ambiguous in meaning. Some other short term may be desirable to take its place in forming these compound names for surface produced. But nothing better has been suggested. "Reciprocal" (of diameter) is too general and too long; "theoretical mesh-tons" is likewise too long and is also liable to misinterpretation; "dia-ton" has been suggested; but it would hardly do to speak of "dia-grams" in connection with a diagram. Perhaps "pitch-grams," "pitch-tons," etc., would be satisfactory, pitch being defined as the theoretical mesh or reciprocal of diameter (probably in inches). I would here again point out that these quantities are measures of surface, and that the weight element is only permitted to enter on the basis of equal specific gravity; where the specific gravity varies, a correction must be made for it.

8. Again I quote from my original paper my interpretation of Kick's law, Stadler's statement of which I have given above:

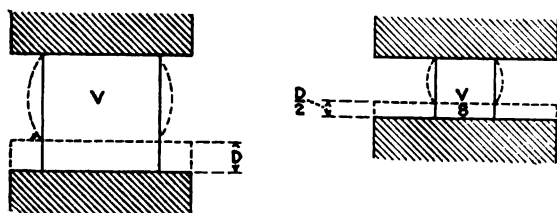


FIG. 6.—KICK'S LAW.

100-Mesh Cube

Area one face = A

Average resisting force per sq. in. = F

Energy = FAD

200-Mesh' Cube

Area one face = $\frac{A}{4}$

Energy = $F \frac{A}{4} \frac{D}{2} = \frac{FAD}{8}$

∴ Energy Proportional to Volume.

"This law does not apply so much to crushing as to deformation of bodies before rupture takes place.

"In Fig. 6 are represented two particles of ore 'of equal technological state,' shown as cubes between the faces of a crushing or testing machine. Assuming the theoretical mesh, equivalent to the reciprocal of diameter, and using concrete values, we have a 100-mesh particle with eight times the volume of the 200-mesh particle, and with an area per face four times that of the 200-mesh particle. The dimensions are as two to one, and the bodies being similarly deformed within the elastic limit without fracture, the energy that must be applied in each case to produce this deformation is the product of the average resisting force per square inch, the same in both cases, by the area worked against and by the distance through which this average force works. As shown in the figure, in this particular instance the energy absorbed is proportional to volume, and it can be similarly shown for the general case. On the gradual release of the external pressure the energy absorbed is given back to the machine producing the deformation [better, which produced the deformation] and the body returns to its original shape. It should be noted that the body has been deformed only by a gradually increasing pressure, the first increment of deformation not requiring so much pressure as the last. In case the body has been deformed beyond its elastic limit

either the whole mass of particles have been reduced to the molecular state by the freeing of their bonds with adjacent particles, which never happens, or fracture takes place along a few surfaces by the breaking down of some of the weaker bonds, and the particles thus formed are free to resume their original shape in so far as they are not held between the machine surfaces. The energy given up by them is probably used in some sort of lever action in making fracture planes. So the energy absorbed according to Kick's law does not stay in the particles after pressure is released and therefore this does not govern to any great extent the amount of energy absorbed in crushing."

Kick's Law

It is evident to the engineer that Kick's law as explained above is but another form of the law of elastic bodies: namely, that within the "elastic limit" stress is proportional to deformation.

10. There is no incompatibility between the Rittinger theory and the laws of elastic bodies such as glass, steel, rock, etc. Up to the elastic limit, deformation is proportional to stress, and absorbed energy is

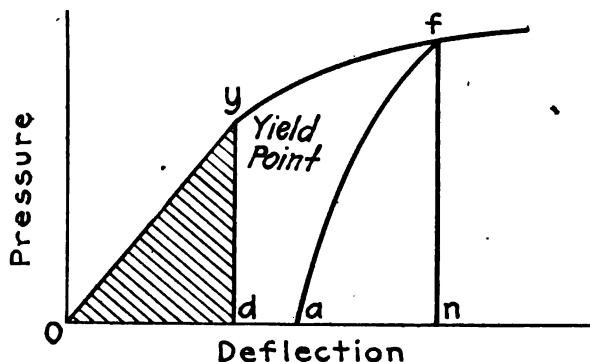


FIG. 7.—STRESS-STRAIN DIAGRAM.

proportional to the volume of the particle stressed, according to Kick's law. In Fig. 7 the area Oyd is a measure of this energy. When pressure within the elastic limit is released the body returns to its former shape, there having been no interior breaks so far. When, however, the elastic limit has been exceeded, certain local ruptures have taken place within the body at points weaker than others; these breaks occur consecutively or simultaneously at various places within the body, and upon release of pressure the body is found upon careful measurement to have taken on a permanent "set." The first fracture noticeable occurs after several ruptures have weakened the body at one point, which may be along a plane of greatest resultant stress, or of structural weakness. Referring back to Fig. 7, the elastic limit having been exceeded, as pressure is gradually released, its relation to the deformation is shown by the line af, the area afn being a measure of the energy given back to the machine

(by Kick's law), while the area *O**f**a* is a measure of the energy absorbed by the local breaks, which may or may not have become visible. In the case of a wrought-iron or mild-steel test bar undergoing tension between the jaws of a testing machine, local ruptures are probably occurring throughout the bar, reducing the section at those points of weakness, and putting greater load upon adjacent particles which may be then able to carry the load better.

The final rupture of the steel bar occurs when a number of local breaks at one particular section have so weakened the bar that all the work the machine is doing is concentrated at that section, as shown by the "necking in" just before failure takes place. Unquestionably in spite of the permanent set in the test specimen away from the point of failure, the majority of the crystals composing the specimen are in the

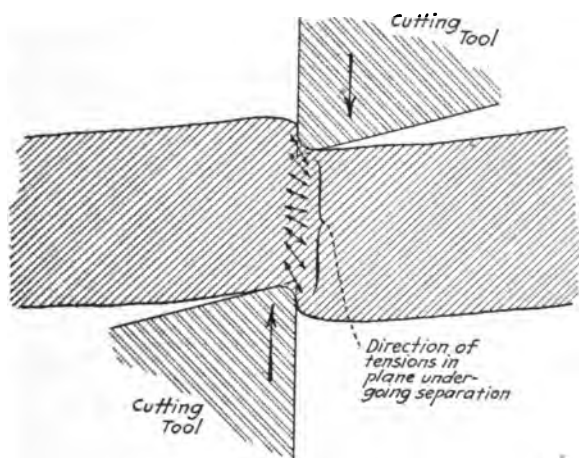


FIG. 8.—SHEARING METAL BAR IN COMMERCIAL MACHINE.

same form as before the test, but some have slipped away from others and there is a new adjustment due to the failure of the few which could not hold on, the others at that section being in position to support the load for a further period. If it were wrought iron, we could say the fiber slipped along the slag or impurities between fibers until the slipping fibers were crowded so that they obtained a sort of "strangle hold."

10. In the case of the shearing of iron and steel bars, it has been shown⁷ that the depth of penetration (the distance traversed by the cutter before separation takes place) is not proportional to the thickness of the metal, as it would be in case Kick's law applied, but is proportionally greater in thin sheets than thick—about 75 per cent. of the thickness in the case of sheets $\frac{1}{8}$ in. thick, and about 25 per cent. in the case of sheets 1 in. thick.

⁷ *American Machinist*, 1905; see references in J. D. Hoffman's *Machine Design*.

On the face of it, this fact does not uphold the theoretical argument for Rittinger any more than for Kick, but let us look farther.

In a commercial shearing machine we do not get true shear, but a condition shown in Fig. 8. The material is compressed under the blades of the shear, which are themselves somewhat distorted during the operation, and failure is due to tension in a line inclined to the plane of the cutting edges. In the case of a thick plate there would be more deformation of the cutting edges due to the higher resistance of the material, and more of a width concerned than in the case of thin sheets. When a hole is punched in steel, there is a volume of material around it that must be removed for first-class work; this volume is probably full of minute fractures the area of which

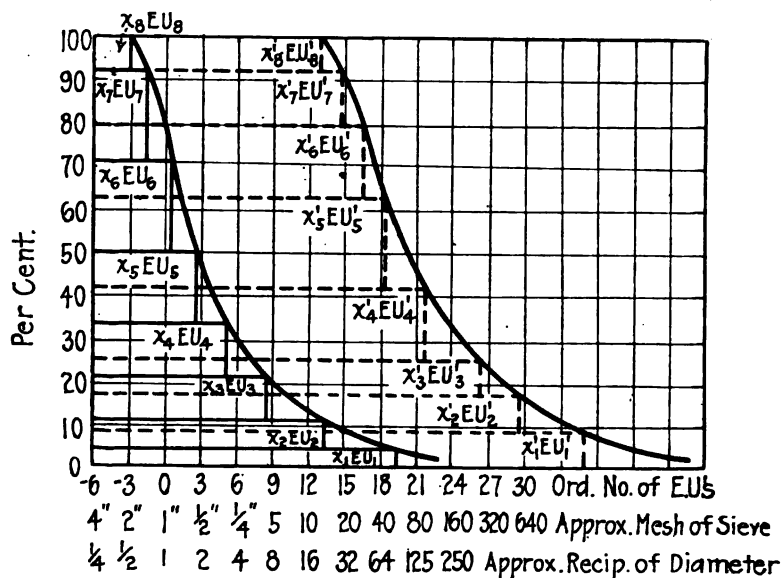


FIG. 9.—MECHANICAL VALUE DIAGRAM.

cannot be measured. I am told that rock crushed for concrete, is seen upon microscopic examination to be full of minute fractures, due to the crushing, which cause each piece to be weaker than a natural pebble of the same mineral and size. If the rock were uniformly full of these invisible fractures, Kick's law would apply in spite of all our theoretical reasoning. Just as in the case of the hole punched in steel, I believe that these minute fractures in the rock are close to the fracture planes.

11. A. F. Taggart, has discussed⁸ Kick's law thoroughly from the standpoint of its advocates and there is no reason for a repetition of his work. However, taking it at their valuation, and using data obtained from Stadler's papers already referred to, we can by reasoning similar to

⁸ The Work of Crushing, *Trans.* xlviii, 153 (1914).

that already employed develop a diagram for energy absorbed similar to the crushing-surface diagram. (Fig. 9.)

With constant weight and similar shapes, Stadler allows one energy unit (E. U.) for each reduction to one-half the volume immediately preceding. Expressed mathematically,

$$\left(\frac{D}{d}\right)^3 = \frac{V}{v} = 2(\text{E. U.})$$

Or, as more frequently expressed,

$$\log V - \log v = (\text{E. U.}) \log 2$$

Whence

$$(\text{E. U.}) = \frac{(\log V - \log v)}{\log 2} = 3 \frac{(\log D - \log d)}{\log 2}$$

D and d being initial and final dimensions, V and v being initial and final volumes. For example: Reducing 1-in. cubes to one-half volume, 1 E. U. has been expended; reducing to one-fourth volume, 2 E. U.; to one-eighth volume, 3 E. U.; to one-fourth dimensions, 6 E. U.; to eighth dimensions, 9 E. U.; and so on. Reducing from 100 in. diameter to 1 in. the same number of E. U. are expended as from 1 in. to 0.01 in., or as from 0.01 in. to 0.0001 in. (all on quantities of the same volume, both here and in the previous sentence). Of course various sizes are produced in crushing, and each grade or size has a different value, the same reasoning applying as in section 4, above.

Suppose mechanical value W be made up of x_1 per cent. of E. U.₁, x_2 per cent. of E. U.₂, etc. x_n per cent. of E. U._n; then

$W = c \sum_1^n x \text{ E. U.}$, before the crushing operation, and

$W' = c \sum_1^n x' \text{ E. U'}$. after crushing.

The new mechanical value, $W - W' = c \sum_1^n (x \text{ E. U.} - x' \text{ E. U'})$.

Plotting x against E. U., we have, when the limiting curves are drawn, a diagram (see Fig. 10) similar to the crushing-surface diagram, the area of which is a measure of the mechanical value of the crushing operation or the energy units (E. U.) absorbed. (Percentages should be expressed decimally in the above calculations.) The mechanical value of sands given in E. U. from the diagram is multiplied by the tonnage per day (or hour) and divided by the horsepower, giving the *relative mechanical efficiency*.

12. The E. U. per foot-pound applied upon 100 g. as found from these mechanical-value diagrams compared with mesh-grams per foot-pound as found from crushing-surface diagrams for the same crushing operation will, if covering a range of operations show which of the two laws under consideration is consistent with the facts. This is the method used in

comparing my results. (In Fig. 9, it will be noted that the theoretical mesh or reciprocals of diameter are designated at the corresponding "Ordinal numbers of energy units," which is self-explanatory, and perhaps permits easier comparison with the crushing-surface diagram for the same crushing operation.) The following comparisons based upon the experimental work performed show that the Rittinger theory applies to crushing operations and that Kick's does not. There are discrepancies when we apply the Rittinger theory; but these can be explained reasonably by considering the expenditure of energy upon the sizes finer than our screen permits our measuring, which must present considerable

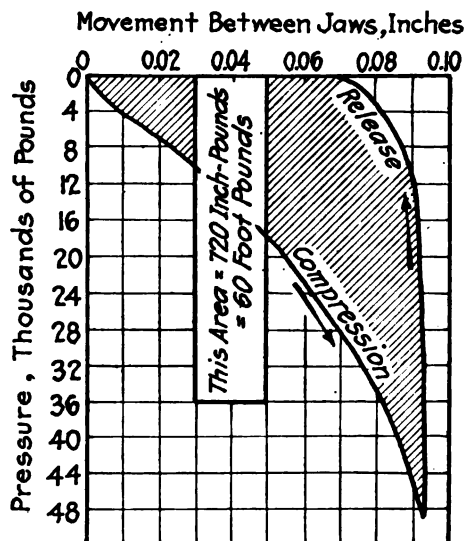


FIG. 10.—ENERGY DIAGRAM.

surface, if we assume that the curve of the screen analysis continues to follow the same law into the finer sizes as among those already measured.

METHOD OF TESTS

13. In this work one or many pieces of rock were compressed between the faces of the Amsler-Laffon hydraulic testing machine in the Laboratory for Testing Materials of Purdue University. (This machine is described in the appendix.) The movement between the faces (similar to those of a crusher) was measured by a deflectometer having a vernier on which movements of 0.001 in. could be read, while the pressures between faces was read upon a dial to a maximum of 150,000 lb. The product of the average pressure (or rock resistance) by the distance through which that pressure was exerted is work or energy, and easily reduced to foot-pounds.

I found it easier to plot my results as obtained, pressure against distance, as shown in Fig 10, the area of the closed curve, completed by the record of pressure and distance when pressure was gradually released, being a measure of the energy in inch-pounds (divided by 12 equals foot-pounds). This energy as foot-pounds, or as horsepower-hours, can be compared with the surface produced as shown by the crushing-surface diagram of the screen analyses of initial and final products of the operation; and it can likewise be compared with the energy units (E. U.) of Stadler, as shown by mechanical-value diagrams similar to Fig. 9. By crushing the same amount of rock from coarse to medium, from medium to fine, from fine to very fine, etc. (coarse and fine being merely relative terms), the two laws may be compared through a sufficient range to detect any abnormal results or inconsistencies.

14. To each test, whether it involved one or several consecutive crushing operations, was given a letter by which it is designated throughout this paper. My records are in the form of crushing-surface diagrams with the weights marked to place as plotted. From these the mechanical-value diagrams must necessarily be derived, as this work was not undertaken primarily to settle the case of Kick vs. Rittinger. The pressure-deformation curves were plotted directly in a similar manner without recording numerical readings. The advantage of recording by plotting the observations directly upon co-ordinate paper is the speed at which it is done, and the check afforded by the shape of the curve. It is hoped that the work will be acceptable in its present form, as numerical tabulations would add nothing of value to this paper. So far as possible the results of calculations are given in the diagrams, and hence it will be unnecessary for the reader to refer back and forth from text to diagram to any great extent.

15. As the crushing-surface diagram does not depend upon any definite system of screens so long as the aperture be known, all the screens used were measured with a Bausch & Lomb microscope having a scale reading to $\frac{1}{1500}$ in., and the reciprocals of diameter of these apertures were used in plotting.*

Certain of the finer sizes were screened through bolting cloth held in embroidery frames. I have suggested specially prepared plotting paper, *i.e.*, with vertical lines drawn in at the reciprocals of diameter corresponding to the apertures of the screens used, for such cases as the Tyler standard screen-scale sieves, or the I. M. M. standard. Since this paper was first submitted to the Institute and of course since the work was done, I have discovered that cumulative direct diagram sheets with Tyler S. S. S.

* In some of the early work actual screen meshes were recorded, but for the sake to uniformity these have been reduced to reciprocals and shown as though recorded that way.

sieves are desirable for this work and that there is no need of any other sheets for crushing-surface diagrams. Fig. 11 shows a crushing-surface diagram on this paper, the plotting running in the opposite direction from that for which it was designed; 1 sq. in. on the regular sheet, with screen sizes located as indicated, representing 8.85 theoretical mesh-tons of surface per ton of ore crushed.

16. All results have been modified to a basis of 100 g., so that percentages required for use with E.U. are readily obtained, about one-third of the tests having been made upon a definite 100 g. In none of the

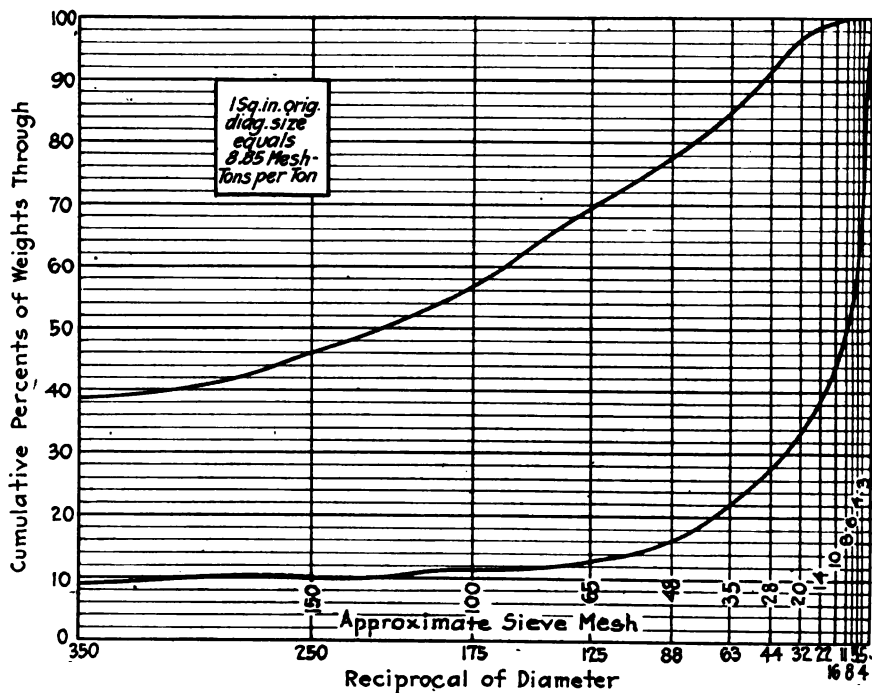


FIG. 11.—CRUSHING-SURFACE DIAGRAM ON TYLER CUMULATIVE DIRECT SHEET.

tests was any attempt made to account for the losses occasioned by fine dust floating away, or by coarser pieces shooting off as they were fractured. In no case could these be more than 2 per cent., which would not materially affect the results. Extreme accuracy was not particularly necessary in this work. It was rather a reconnaissance; and if more than three significant figures are given at any point, too much importance should not be attached to the last figure. At the same time the work was carefully done and the record carefully made, but there is so much unknown that I would not care to defend the fine details of my results.

RECORD OF TESTS

17. In Fig. 12, A and B are results obtained upon a tube-mill feed, a fairly siliceous gold and silver ore from Mexico, of which 100 g. of $-5\frac{1}{8}$ -in. +4-mesh pieces were crushed by two movements of the machine, without

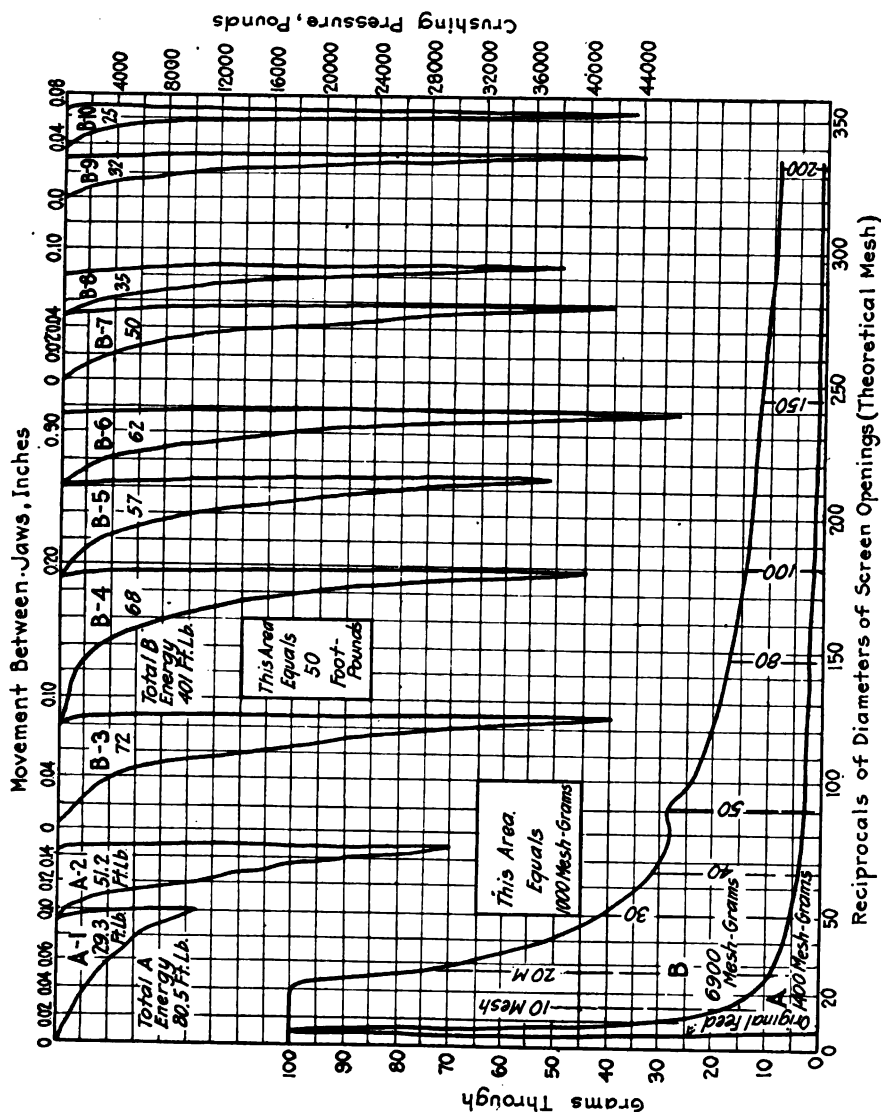


FIG. 12.—CRUSHING-SURFACE AND ENERGY DIAGRAMS. SILICEOUS MEXICAN ORE.

intermediate screening, the purpose being to crack every piece. The energy is represented by the areas A-1 and A-2. It is of interest to note how the release of maximum pressure is usually accompanied by

some movement of the jaw, as in this case, indicating a return of energy to the machine, shown in the diagram by the slight curve back to the zero-pressure line. After the particles adjusted themselves upon release of the pressure it took more movement between the crushing faces to get up to the previous pressure, as indicated by the gap between A-1 and A-2. After sizing and plotting the screen analysis in the form of a crushing-surface diagram A, the particles were re-crushed eight times, each time being screened through a screen of about 30 mesh to remove the fines and eliminate the condition of "choke crushing" as far as possible, the oversize only going back to the machine. The energy areas for these eight crushings are B-3, B-4, B-5, B-6, B-7, B-8, B-9, and B-10; the screen analysis is plotted as a new side of the crushing-surface diagram, B. The areas of the energy diagrams A-1 and A-2 were found by planimeter to be 1.40 and 2.38 sq. in. respectively, a total of 3.78 sq. in. One square inch, for the scales used, being equal to 250 in.-lb. or 20.83 ft.-lb., 3.78 times 20.83 or 80.5 ft.-lb. of energy was expended in making the crushing A. By similar reasoning, the area of the crushing-surface diagram A, as found by planimeter (as far as screening was carried out, to the 200-mesh sieve), multiplied by the value of its unit area in mesh-grams, shows that 1,400 mesh-grams of surface was produced. From

these two quantities, a value of $\frac{1400}{80.5} = 17.3$ mesh-grams per foot-pound

(+200-mesh sieve) was obtained, as the constant to be used in calculating efficiencies, etc., for this rock. Note, however, that the mesh-grams entering into this constant are only those "plus" 200-mesh sieve, and that there are perhaps more mesh-grams in the "minus" 200-mesh than in the "plus," and they must at some time be taken care of in calculations. It is these "minus" mesh-grams which will account for the discrepancies in results by Rittinger's theory, and which will not account for discrepancies in results by Kick's law.

18. For B, the total energy expended was 401 ft.-lb., producing 6,900 mesh-grams (+200-mesh sieve) giving a unit value of 17.21 mesh-grams per foot-pound "plus." Fines were screened out after each jaw movement, to reduce "choke crushing."

19. Tests E, F, G, H, J, K, L, M, and N were made upon Bedford limestone, the first in July, 1912, and the rest in July and August, 1913. This is one of the best known building stones in the United States. It occurs in great beds in southern Indiana, is oölitic in texture (weathering brings out myriads of fossils) and, because of its apparent uniformity, is probably as good a rock as could be selected for a standard. (I reserve the right to change this opinion later.) The particular specimens came from stone furnished for the Purdue Library building.

20. The crushing-surface diagrams of E and F (Figs. 5 and 4 in my original article in the *Engineering and Mining Journal* of May 24, 1913) are

not shown here. In E, practically all the material was broken in the machine, whereas in F, a great many pieces between 30 and 80 mesh were not broken, but simply had their edges rubbed off. My records as to F

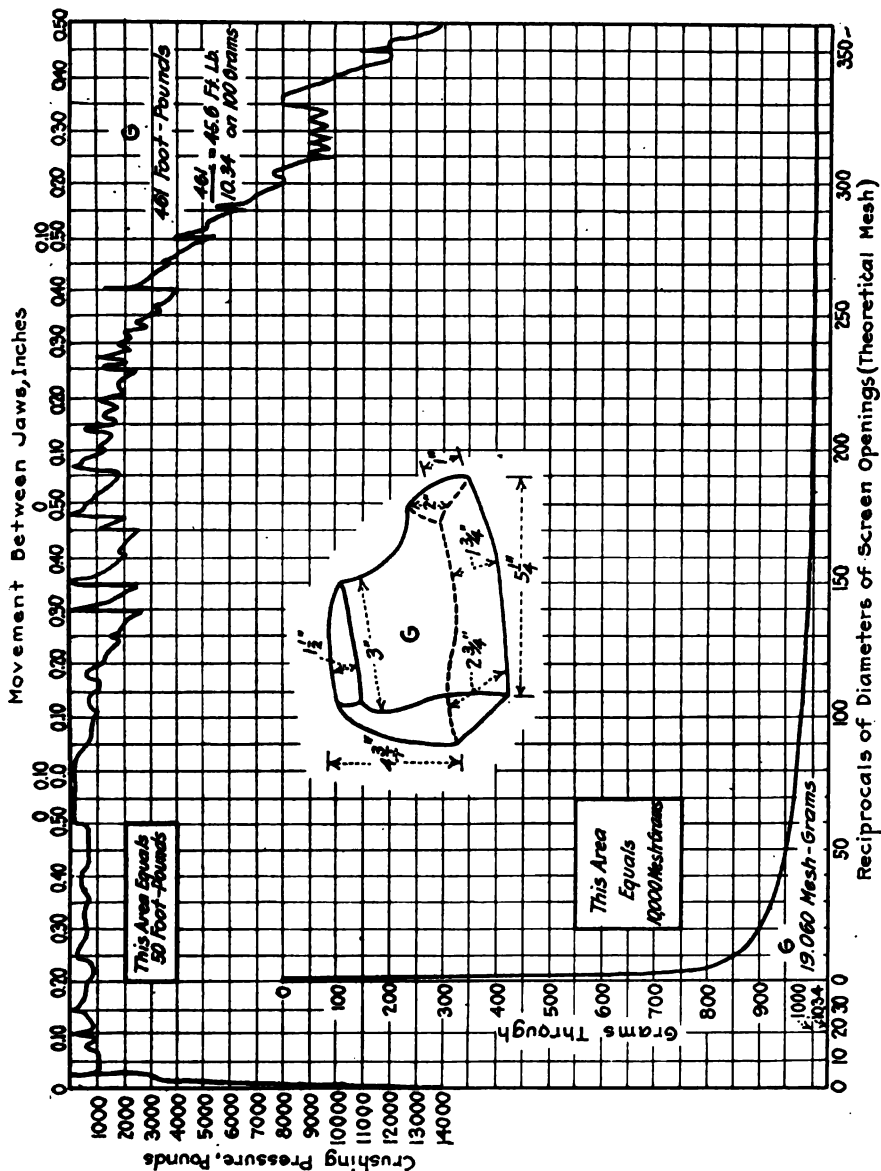


Fig. 13.—CRUSHING-SURFACE AND ENERGY DIAGRAMS, AND FEED-SAMPLE. BEDFORD LIMESTONE.

do not clearly show whether I screened between crushings; this was done in the case of E.

21. Test G was on a block of the dimensions shown in the small

sketch in Fig. 13, having a superficial area of 63 sq. in. and a weight of approximately 1,031 g. The energy curve is rather long drawn out after the first break, and the irregularities in this curve may be explained

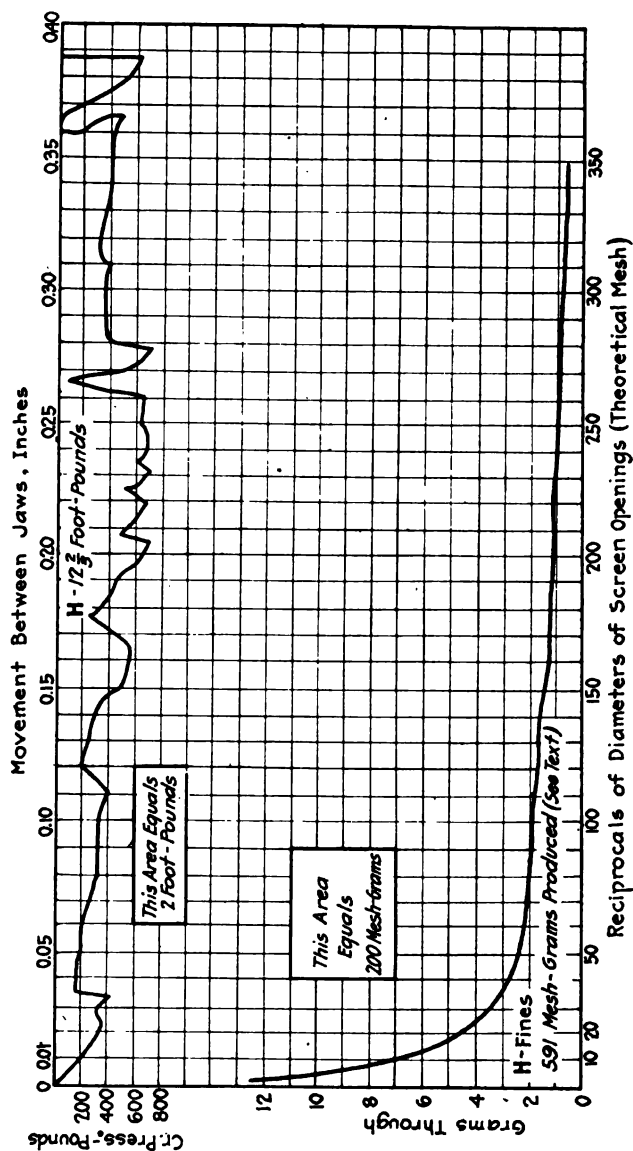


FIG. 14.—CRUSHING-SURFACE AND ENERGY DIAGRAMS. BEDFORD LIMESTONE.

as due to frequent ruptures of fairly large pieces where the load drops back almost to zero. A total movement of nearly 2 in. is recorded: here only one or two pieces perhaps were resisting, while in some of the other

cases a great many pieces were resisting at one time. At the end of the first $\frac{1}{2}$ in. of movement the finer pieces were screened out, and the larger returned to the crushing zone; these pieces being rather flat; about 2 per cent. of -200 mesh being produced.

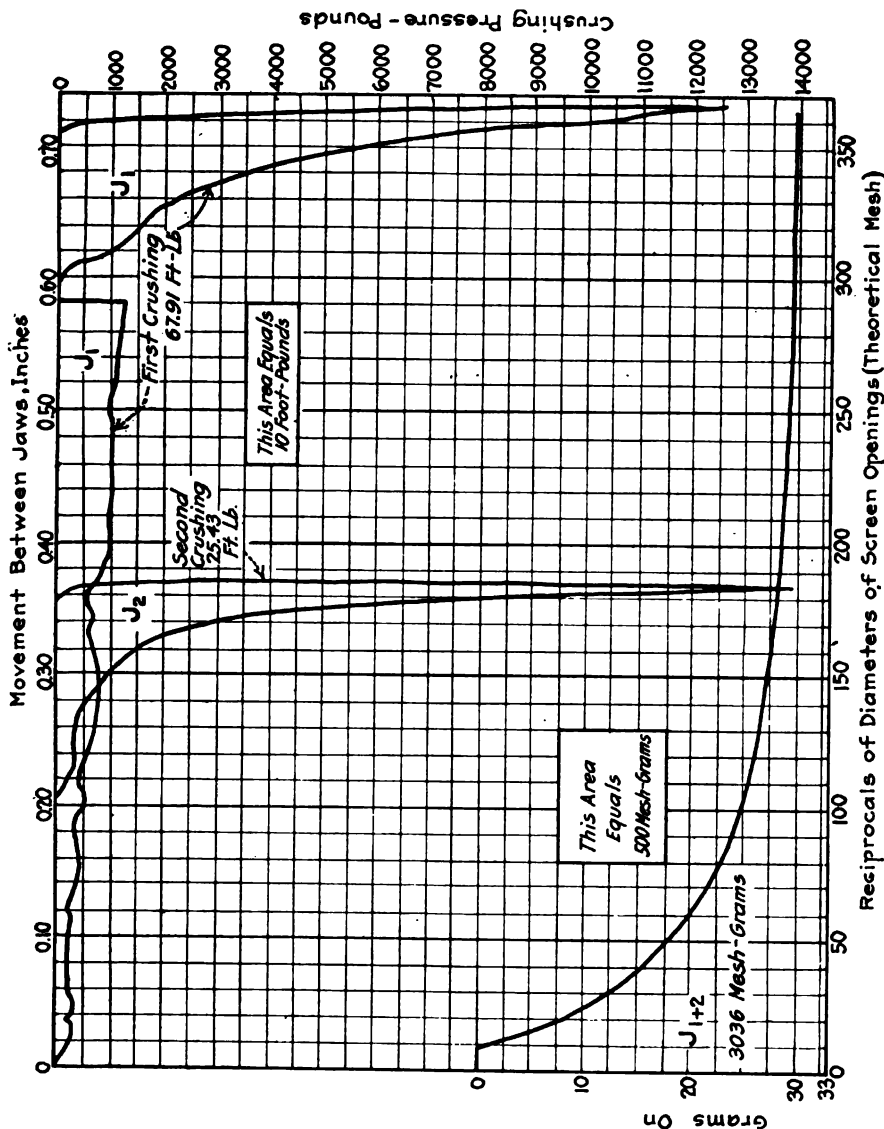


Fig. 15.—CRUSHING-SURFACE AND ENERGY DIAGRAMS. BEDFORD LIMESTONE.

22. H, shown in Fig. 14, covers the results of crushing four pieces having a total superficial area of 30.25 sq. in. and a total weight of 175.5 g. After crushing, eight pieces weighing 163 g. had an area of 30.25 sq. in., just offsetting the original area, while the remaining 12.5 g.

were screened, and the results were plotted as a crushing-surface diagram. 12.66 ft.-lb. were expended in crushing the 175.5 g., equivalent to 7.2

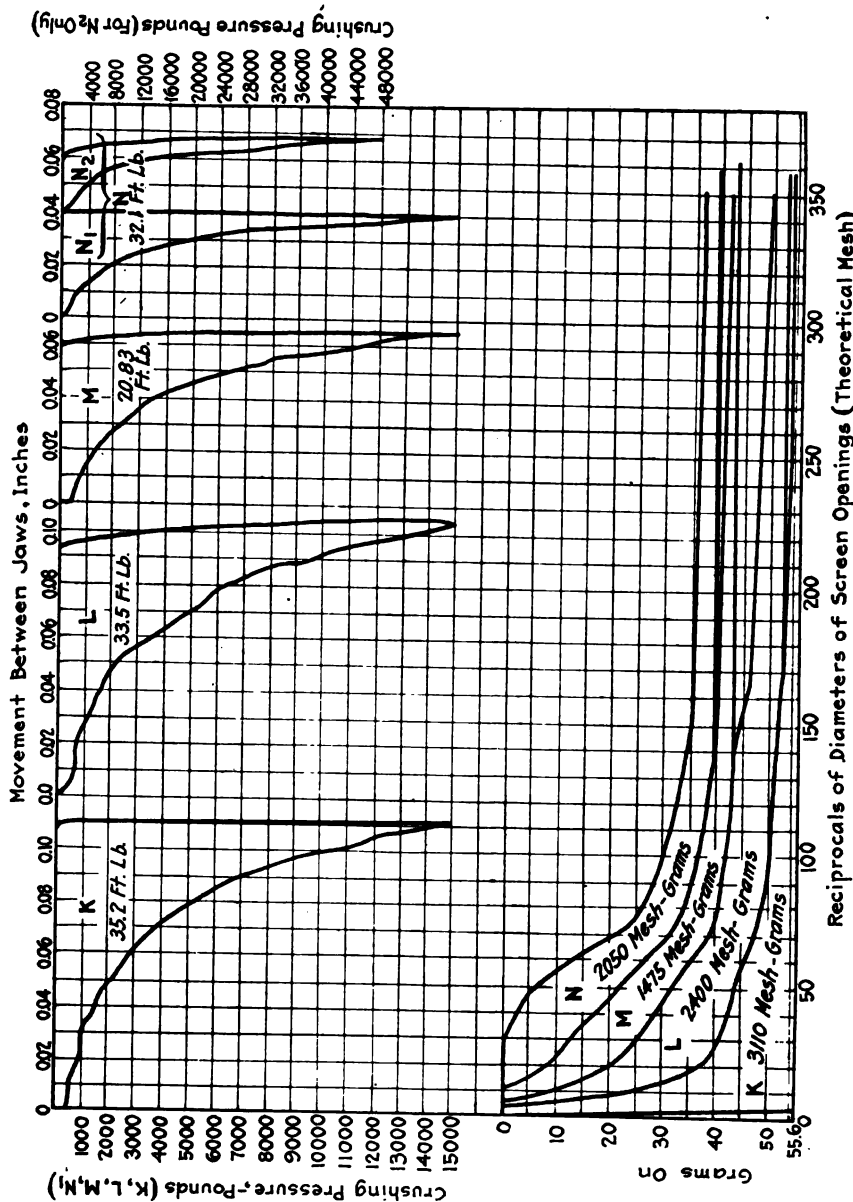
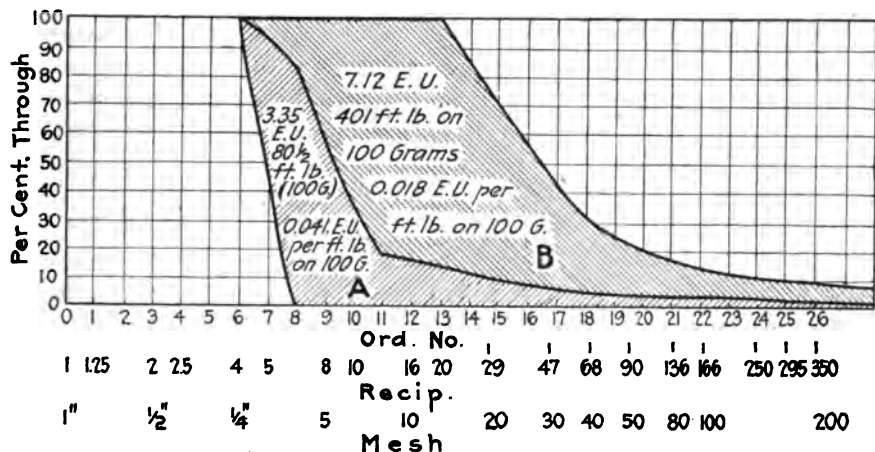


Fig. 16.—CRUSHING-SURFACE AND ENERGY DIAGRAMS. BEDFORD LIMESTONES.

ft.-lb. per 100 g.; the eight pieces noted above were estimated to be the equivalent of 60 per cent. of 1 in., 32.5 of $\frac{1}{2}$ in., and 7.5 of fines, as recorded in the crushing-surface diagram. The energy curve is irregular as

in the case of G and for the same reason, movement was 0.4 in.; there was no intermediate screening.

23. In J (Fig. 15) a single rectangular piece about 1 by 1.25 by 1.25 in., weighing 33 g., was crushed to pass a 10-mesh sieve, with one intermediate screening. After a deformation of about 0.6 in. was recorded



[FIG. 17.—MECHANICAL VALUE DIAGRAM. SILICEOUS MEXICAN ORE.

without great pressure, the pressure goes up very rapidly, probably by reason of the packed condition of the particles, a great number resisting crushing. After screening out the -10 mesh, a similar energy curve is obtained on the +10-mesh pieces. The crushing-surface diagram represents the combined product from both crushings. In both cases, the

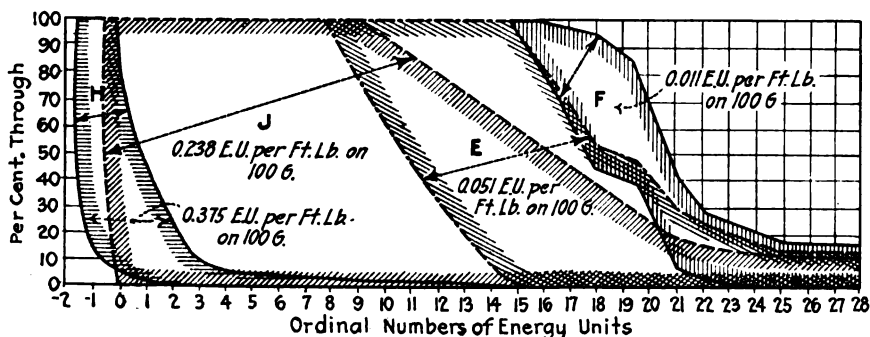


FIG. 18.—MECHANICAL VALUE DIAGRAMS. BEDFORD LIMESTONE.

packed condition, in which the particles rub on one another and produce much -200 mesh, accounts for the low value of the mesh-grams per foot-pound.

24. Fourteen very flat pieces weighing 55.6 g., with about 20 sq. in. of surface, averaging in size about 0.75 in. sq. by $\frac{1}{8}$ in. thick, were sub-

jected to four separate crushing operations, K, L, M, and N, as shown in Fig. 16. The first crushing K was continued until every piece was found to have been broken at least once; screen analysis was then made and recorded in the crushing-surface diagram, and the material +100 mesh was recrushed as L. M is the result of crushing in a similar manner the +80 mesh from L; and N is similarly the result from the +80 mesh of M. Each crushing was carried as far as the record shows without attempting to remove any fines, although in the case of N pressure was released once and then reapplied, as recorded, in an endeavor to apply more energy. The base lines of L, M, and N separate from the discharge or product lines of the preceding operations, showing the smaller quantities of material crushed in each case.

25. In Fig. 17 the results obtained in A and B on the siliceous Mexican ore have been plotted in a mechanical-value diagram. For the coarser crushing A, 0.041 E. U. per foot-pound-applied on 100 g. was the result,

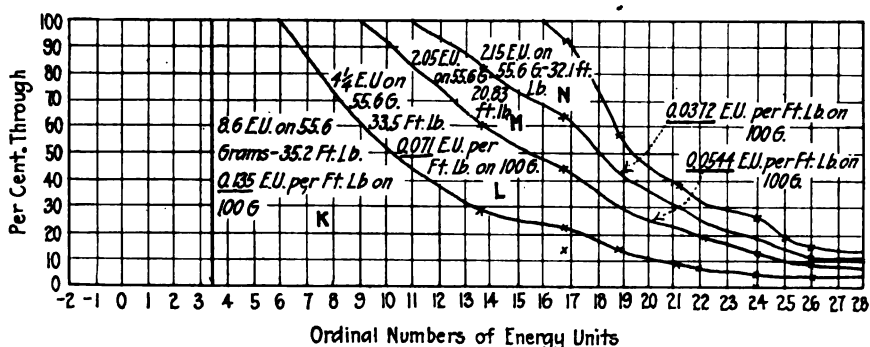


FIG. 19.—MECHANICAL VALUE DIAGRAMS. BEDFORD LIMESTONE.

while in the case of B only 0.018 E. U. (etc.) resulted, 125 per cent. more being produced when the coarser crushing was done. If graduated crushing had not been done, more energy would have been expended upon the -200 mesh; and I believe this discrepancy would then be greater.

26. In Fig. 18, E, F, H, and J (Bedford limestone) have been plotted as mechanical-value diagrams. Note how the E. U. per foot-pound applied on 100 g. decrease consistently from coarse to fine: H, 0.375; J, 0.238; E, 0.051; and F, 0.011. Similarly in Fig. 19, K, L, M, and N are plotted to E. U. diagrams with results decreasing consistently from coarse to fine: K, 0.135; L, 0.071; M, 0.0544; and N, 0.0372 E. U. per foot-pound-applied on 100 g.

27. On another ore, an Arizona porphyry, a series of crushings similar to K, L, M, and N, gave results from coarse to fine as follows: O, 0.055; P, 0.049; Q, 0.0296; R, 0.0149; S, 0.0127; all E. U. per foot-pound-applied on 100 g. The range of sizes was small; the initial feed averaged

TABLE I.—*Summary of Results*

Character of Rock	Mexican Siliceous Ore ¼ in.		Bedford Limestone										Porphry, Mill Feed					
	A	B	E	F	G	H	J	K	L	M	N	O	P	Q	R	S		
Operation.....																		
Mesh-Grams of Surface Produced (Plus 350 Reciprocal).....	1,400	6,900	8,800	8,450	19,060	665	3,036	3,110	2,400	1,475	2,050	1,250	1,225	1,470	1,254	1,220		
Foot-Pounds Energy Applied.....	80.5	401	166.3	295.6	461	12.7	93.3	35.2	33.5	20.8	32.1	50	48	564	103.3	75		
Mesh-Grams per Foot-Pound (Plus 350 Reciprocal).....	17.3	17.21	53	26.7	41.3	63.5	33.6	88.4	71.9	70.3	63.3	25	25.5	26.1	13.16	16.3		
Stadler's E. U.'s per Foot-Pound Applied on 100 g....	0.041	0.018	0.051	0.011	0.213	0.375	0.238	0.135	0.071	0.0544	0.0372	0.055	0.049	0.0296	0.0149	0.0127		
Remarks.....	Series Coarse		Medium		Fine		Large Block		Coarse Pieces		Choke Crushing		Series Medium		Fine Choke		Crushing	

about 4 mesh, while all five crushings resulted in the production of not more than 8 per cent. of -200 -mesh sieve product. This work was done in December, 1913.

28. Fig. 20 shows the results of plotting an approximate average of feed and discharge for all tests against both the E. U. per foot-pound-applied on 100 g. and the mesh-grams per foot-pound ($+350$ reciprocal).

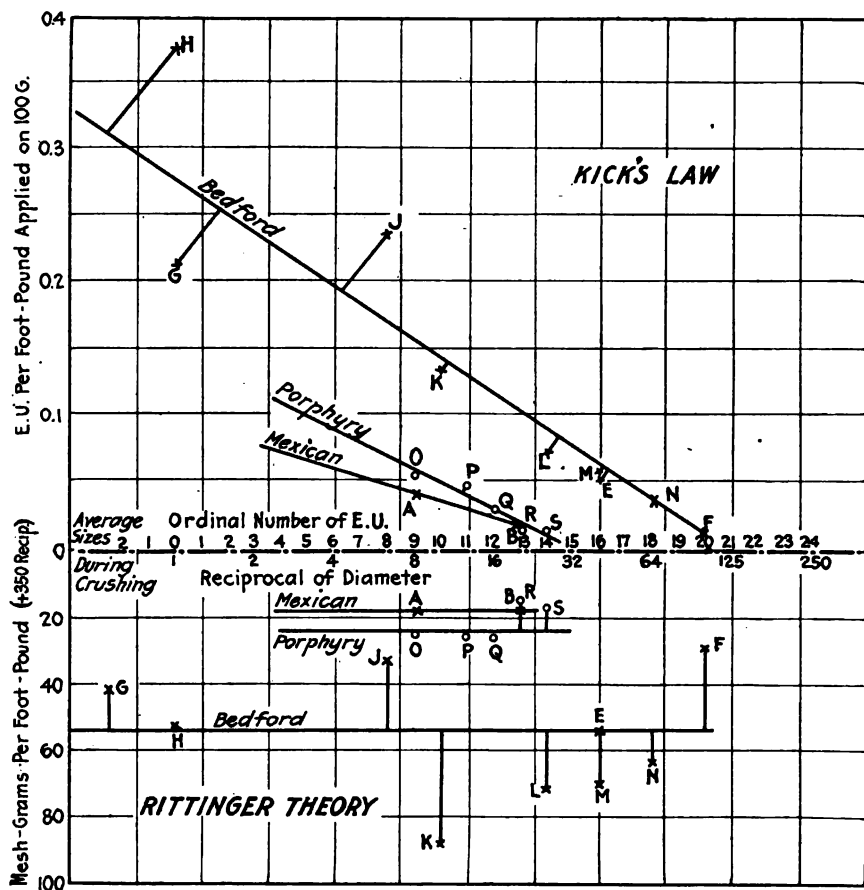


FIG. 20.—VARIATION OF TEST RESULTS BY BOTH METHODS.

The estimated average, taken as near to the center of gravity of the mechanical-value diagrams as possible by ocular observation, gave the horizontal position of each point, while the calculated E. U. and mesh-grams gave the two vertical positions.

In the upper diagram the inclined line (Bedford limestone) connecting the various points shows that for different average sizes of material undergoing crushing, the production of E. U. per foot-pound-applied on 100 g.

decreases from coarse to fine; according to Stadler's interpretation of Kick's law there should be no decrease, they should be the same at all sizes. From the values tabulated in Table I, which is a summary of all these tests and calculated results, it will be found that for the Bedford limestone, the *coarsest* crushing gave 34 times the number of E. U. per foot-pound-applied on 100 g. that the *finest* crushing gave. For the other materials, the graphs incline similarly and in sufficient degree to show the fallacy of Stadler's application of Kick's law.

29. In the lower diagram, the results of this work on a Rittinger basis have been plotted, adhering, however, to the average sizes, taken as described above, which will not affect the results, except to lengthen the curve in the coarse sizes and to shorten it in the fine sizes. Horizontal straight lines averaging the results for each material have been drawn in; if the same number of foot-pounds-applied produce the same number of mesh-grams at all sizes, then the Rittinger theory is correct, and a horizontal line represents it in this diagram. If some of the discrepancies be explained as due to "choke crushing," producing an excess of unmeasurable -200 mesh (-350 reciprocal), the majority of points will come nearer to a horizontal line, and better confirm our belief in the Rittinger theory. The results clearly show that the Rittinger theory is so much more nearly correct for crushing operations, that I believe we can accept it and use it for the determination of crushing constants, for machine comparisons, for efficiency tests, etc.

30. Since it is obvious that the sizes finer than our ordinary screens will separate are important if the Rittinger theory be correct, finer screens or classification followed by microscopic measurements will probably be necessary for accurate determinations. To some extent at least, we can approximate the quantity of the finer sizes; there seems to be a law connecting the quantities of different sizes, that the discharge line of a crushing-surface diagram from a uniform crushing operation will be in the form of a hyperbola, and, when plotted on logarithmic paper, as in Fig. 21, will appear as a straight line. In Fig. 21 the product line of N indicates that it would have some 4 per cent. of 1,000-reciprocal size and some 0.6 per cent. of 10,000-reciprocal size; they have surfaces respectively equivalent to 40 per cent. of 100 reciprocal, and 60 per cent. of 100 reciprocal, taken separately; and if these values are correct, these sizes are fully as important in the consideration of energy disposal as all of the +100-reciprocal material. The law of the finer sizes might be something of this form:

$$RW^x = C$$

R being the reciprocal of the diameter (theoretical mesh) under consideration; W, the weight of the material passing through a screen of this diameter opening; and x, a constant depending upon the character of the

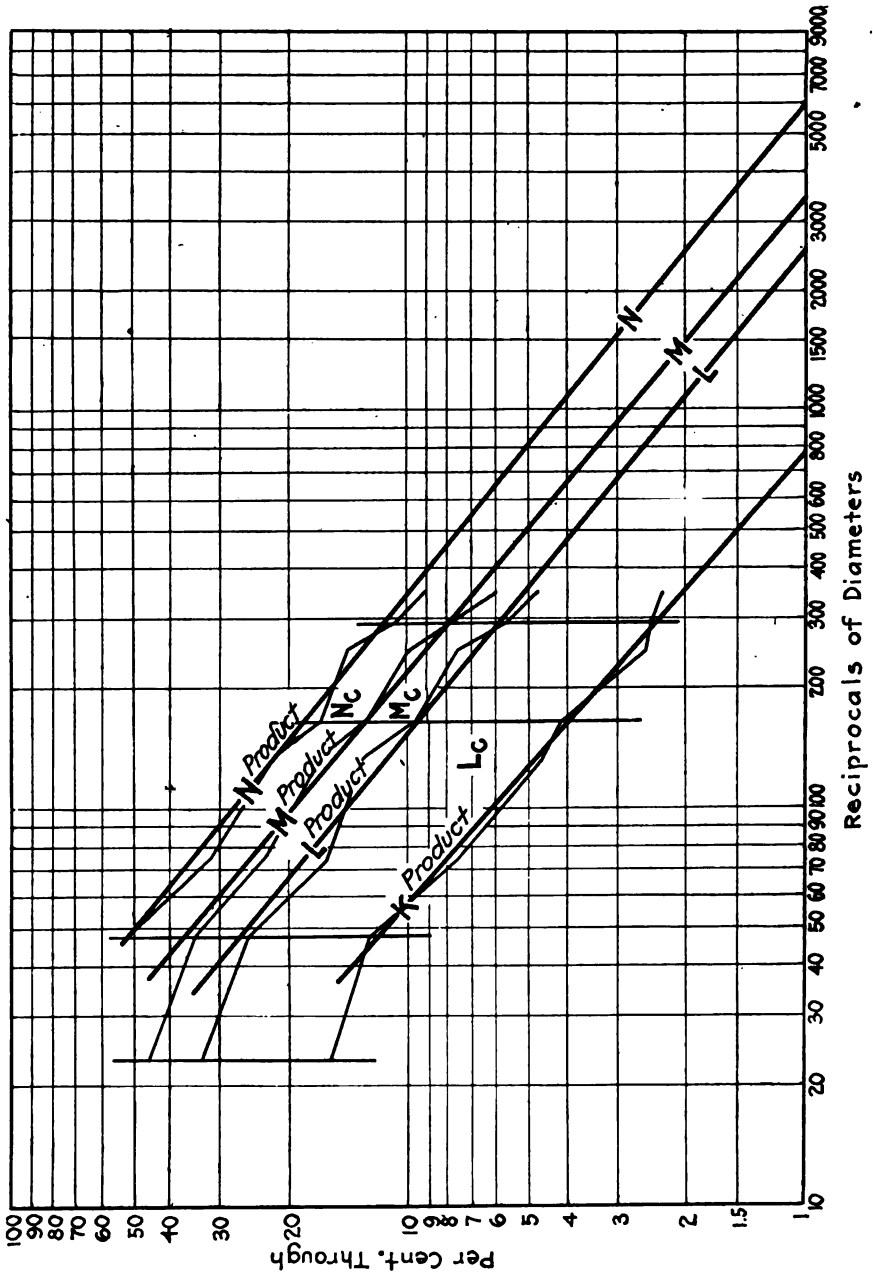


Fig. 21.—SCREEN ANALYSES PLOTTED TO LOGARITHMS.

ore, the type of machine in which crushing is done, and possibly upon the screen analysis of the feed; and C, the number of crushing units (mesh-grams, mesh-tons, etc.) in the undersize, considered as all of that one size, and proportional to the area between the co-ordinates of the point on the curve (intersection of C and W) and the zero lines.

This, however, is largely speculative, and is only suggested because it may be of some value to some future investigator.

SUMMARY OF RESULTS

31. Table I gives a summary of results. It will be noted that in most cases the high values for mesh-grams per foot-pound were obtained with low values in foot-pounds, and *vice versa*. This seems to indicate that the highest values and therefore the most efficient crushing would be obtained when each particle was just cracked; a thing that is generally understood to be true in connection with rolls, for example, free crushing being, within limitations, more efficient than choke crushing. Choke crushing here gave uniformly low results, as might be expected. In the case of B, the several crushing operations involved small amounts of power, but they were lumped together for final screen analysis. If all the power had been applied at one time, without intermediate screening, there would not have been such a nice agreement in the mesh-grams per foot-pound, between A and B.

32. As the writer sees the situation, after the acceptance of the substantially correct theory of crushing, there must be recognition of the fact that rocks of varying characteristics are met in crushing operations, that these rocks offer different resistances to crushing, and that it is unfair to compare the operations of two crushing machines upon dissimilar ores (even if they look alike) until some values are given to the resistance or energy absorption of these rocks under crushing conditions. Probably these units will be determined practically by applying a definite amount of energy to a definite weight of the rock, all of a predetermined standard size. The work of determining these constants should be done thoroughly by competent hands. The matter of checking my conclusions can be accomplished perhaps in several of the school testing laboratories of the country. No elaborate apparatus, and apparently no very high degree of accuracy, will be required to prove that Kick's law is of no use to mill operators and engineers.

33. This paper has been prepared under some disadvantages which may account for the possible lack of logical arrangement. The matter was originally submitted to the committee of the Institute in May, 1914, and upon their suggestion the paper was enlarged, corrected, and rearranged. The writer acknowledges his indebtedness to their suggestions. Acknowledgment should also be made of the courtesies extended by



FIG. 22.—AMSLER-LAFFON TESTING MACHINE.

Dr. W. K. Hatt, head of the department of Civil Engineering, and Professor Schofield, at Purdue University, who allowed the writer free run of the laboratories for Testing Materials.

APPENDIX

The Amsler-Laffon hydraulic testing machine, with which these experiments were made, is shown in Fig. 22.

The material is crushed between the faces of the jaws *J* by pressure applied against the ram *R*, contained in the cylinder *C*. The head *H* carrying the upper jaw can be moved up and down the fixed screws *S* by nuts having worm-wheel threads cut on their outer surface into which mesh two worms driven by the hand crank shown to the left.

The pump *P* for supplying the pressure to the ram cylinder *C* is operated by a hand screw on the side of the frame carrying the registering and auxiliary mechanism. Suitable piping connects the pump with the cylinder, the supply tank *O* and the registering cylinder *X*. From *X* suitable links and levers connect to the pendulum and adjustable weight *W*, the swing of the pendulum, which is proportional to pressure exerted, being recorded on the drum or indicated upon a dial *D*. In all of this work the readings of the dial were used. With different settings of the weight *W*, four different scales of pressures having maximum values of 15,000, 50,000, 100,000 and 150,000 lb. may be used, suitable dials being provided.

The deflectometer, which is a part of the regular laboratory equipment, consists of a lever carried at its fulcrum in a suitable base. At the short end of the lever a fine-thread vertical adjusting screw can be brought into contact with the upper jaw *J* of the machine, while the long end of the lever fits against a vertical scale, and moves a vernier with it as it moves up and down, thus permitting the measurement to 0.001 in. of the movement between jaws.



DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Standardizing Rock Crushing Tests

BY MYRON K. RODGERS,* B. S., LOS ANGELES, CAL.

(San Francisco Meeting, September, 1915)

IN rock- or ore-crushing tests all data, in order to be valuable for study and comparison, should be obtained and tabulated under conditions as uniform as possible. The results of many such tests have no value for comparison, because of incomplete data and the lack of uniform conditions, screens and scale of sizing, etc.

The purpose of this paper is to suggest a form of standard specifications which may be developed by the Milling Committee and approved by the A. I. M. E. for rock-crushing tests. It is not intended to discuss any theories of rock crushing.

Reports of rock-crushing tests should include the following details:

1. Description of the machine employed (jaw or gyratory crushers, rolls, stamps, tube mills, Chilean mills, etc.).
2. Method and material (timber, concrete, etc.) of foundation. Much power is dissipated in the vibration of poor foundations.
3. Locality from which the rock or ore was obtained, and geological, mineralogical, and physical characters of the material.
4. The power consumption of the machine running with no load and with full load, the unit of power being 1 hp. per 24 hr.
5. The capacity of the machine in tons (of 2000 lb.) per 24 hr.
6. The duty in tons per horsepower-day.
7. The screen analysis of feed and product by the proposed A. I. M. E. standard screen scale.

On the foregoing statement, the following remarks are offered.

Toughness of Rock

Table I shows the relative crushing duty of the same machine on three different characters of rock, all other crushing conditions being the same.

This table indicates that the specific resistance of the rock to crushing, may affect the crushing duty of a unit of power as much as 300 per cent. Perhaps this quality may be called toughness. It is not a function of hardness only. For comparison of tests, rocks should be divided into

*Mining Engineer.

TABLE I.—*Relative Crushing Duty of the same Rock-crushing Machine on Three Different Characters of Rock, all Other Crushing Conditions Being the Same*

Rock and Feed	Mill	Per Cent. Solids	H.p. of Machine	Tons per Machine, 24 hr.	Tons per Hp.-Day	Tons 200 Mesh per Hp.-Day	Screen Scale +4+10+20+40+60+ 100+200-200									
Calumet & Hecla conglomerate, ¼ mesh.	Conical tube 8 ft. by 30 in.	40	50.0	65	1.30	0.44	Feed									
							0	65	29	5	1	..	..	..	..	..
							Product									
							..	..	..	3	9	13	41	34		
Cœur d'Alène quartzite, F. M. & S. Co., 5 mm.	Conical tube 8 ft by 22 in. 28 r.p.m.	43	47.5	112	2.50	0.67	Feed									
							0	43	43	11	1	1	1	1	1	1
							Product									
							..	..	2	22	18	19	12	27		
Miami, ¼ mesh.	Conical tube 8 ft. by 22 in.	27	47.5	229	4.85	1.80	Feed									
							4	45	33	10	2	2	1	7	7	7
							Product									
							..	..	2	14	12	17	14	38		

Relative crushing duty of 1 hp. due to character of ore only:

Calumet & Hecla Conglomerate.....	1.00
Cœur d'Alène Quartzite.....	1.92
Miami.....	3.75

at least three classes in this respect; and the class to which a particular rock belongs might be determined by its crushing strength per square inch.

Screens

For screen analysis, a standard screen scale should be used, having a uniform ratio between the different sizes.

The results of an investigation made by the U. S. Bureau of Standards, with several of the large mining companies, where 240- to 260-mesh screens are used in connection with flotation tests, and including several cement manufacturers and screen manufacturers, are that a screen analysis for all rock-crushing tests, and practically all testing of materials, will be covered by a screen scale based on the U. S. Bureau of Standards, 200-mesh screen with a square opening of 0.0029 in. and a wire of 0.0021 in. diameter, advancing with a ratio of 1.414 (*i.e.*, the square root of 2, suggested by Rittinger) between the successive sizes of screens, to a square opening of 4.197 in. area corresponding to a 4-mesh screen; and where finer than 200 mesh is required, a 280 mesh or 0.0020 opening could be used, retaining the above ratio.

TABLE II.—*Standard Screen Scale*

Mesh per Linear Inch	Opening, Inches	Opening, Millimeters	Diameter of Wire, Inches	Diameter of Wire, Millimeters	Area of Openings, Square Inches
	4.20	106.60	0.375	9.52	17.64
	2.97	75.39	0.207	5.26	8.82
	2.10	53.33	0.192	4.88	4.41
	1.49	37.73	0.149	3.78	2.20
	1.05	26.67	0.149	3.78	1.10
	0.742	18.85	0.135	3.43	0.551
	0.525	13.33	0.105	2.67	0.276
	0.371	9.423	0.092	2.34	0.138
3	0.263	6.680	0.070	1.78	0.069
4	0.185	4.699	0.065	1.65	0.034
6	0.131	3.327	0.036	0.91	0.017
8	0.093	2.362	0.032	0.81	0.0086
10	0.065	1.651	0.035	0.89	0.0042
14	0.046	1.168	0.025	0.64	0.0021
20	0.0328	0.833	0.0172	0.44	0.00108
28	0.0232	0.589	0.0125	0.32	0.00054
35	0.0164	0.417	0.0122	0.31	0.00027
48	0.0116	0.295	0.0092	0.23	0.000135
65	0.0082	0.208	0.0072	0.18	0.0000672
100	0.0058	0.147	0.0042	0.11	0.0000336
150	0.0041	0.104	0.0026	0.07	0.0000168
200	0.0029	0.074	0.0021	0.05	0.0000084

If the analysis is to be carried finer than 0.0029 in. (200 mesh), the next finer sieve-opening in the Screen Scale Series, is 0.002 in. (280 mesh).

The writer is under obligations for practical assistance in the preparation of this paper to W. S. Stratton, Director, Rudolph J. Wig, and R. Y. Turner, of the U. S. Bureau of Standards; Frederick Laist, Metallurgical Engineer, and Albert E. Wiggin, Superintendent of Concentration, of the Anaconda Copper Mining Co.,; and A. S. Disbro of the W. S. Tyler Co., Cleveland, Ohio. Mr. Disbro has practically developed the suggested screen scale.



The Rôle and Fate of the Connate Water in Oil and Gas Sands

Continued discussion of the paper of ROSWELL H. JOHNSON, presented at the New York meeting, February, 1915, and printed in *Bulletin* No. 98, February, 1915, pp. 221 to 226. See also *Bulletin* No. 101, May, 1915, pp. 1157 to 1162, and No. 103, July, 1915, pp. 1449 to 1459.

C. W. WASHBURN, New York, N. Y. (communication to the Secretary*).—Mr. Johnson recognizes that connate waters must be driven outward toward the earth's surface. He considers the ascent due to the settling and packing of the underlying sediments and to the formation of gas within the rocks, supplemented by abyssal pressures.

The origin of the methane in the sedimentary strata probably is not simple. There are three possible modes of origin: (1) as marsh gas formed near the surface and entrapped in the sediments; (2) from abyssal sources; (3) by the decomposition of organic matter buried in the sediments. The abyssal source is generally negligible, and in all cases doubtful. The third origin is the most important in the gas fields. A fourth mode of origin, by the dissociation of liquid hydrocarbons, frequently has been suggested, but it is improbable, because crude oils are exothermic. Therefore they have no tendency to split or decompose at low temperatures, as do endothermic substances, such as coal, which is altered geologically by the loss of methane, water, and carbon dioxide. This cannot happen to a hydrocarbon until its temperature of dissociation ("cracking point") is reached, which is over 250° C. for the most easily cracked hydrocarbons and over 400° C. for others.

Engler says that the dissociation of fats and waxes into oils is an endothermic process, thereby unconsciously nullifying his own theory of the dissociation of these substances at low temperatures. However, recently I have found reason to think that heat is liberated by the disruption of the molecules of fats and waxes, as it is in the molecular decomposition of the substances of wood, coal, etc. If this tentative opinion is correct, and Engler wrong, then it seems that the Engler-Höfer hypothesis of the origin of oil becomes not only possible, but even probable.

A second conclusion is that all complex organic molecules which remain in their original state to the extent of retaining combined oxygen must have a tendency to decompose, with the production of exothermic substances, including methane and other hydrocarbons. This would result in pressure, since the products occupy greater volume than the parent substances. To reverse the problem, as in the paper, and to say that pressure itself would cause the production of methane from organic

* Received July 6, 1915.

matter, is a confusion of cause and effect, and seems to be contrary to physical principles. Those who appeal to pressure in this way probably are overlooking the essential factors. Since the investigations of Brame and Cowan¹ and of others have shown that coal is endothermic, its natural underground alteration, even at low temperatures, is explained readily by the common principles of thermo-chemistry. The expression "dynamo-chemical stage" of alteration could be replaced advantageously by the better established word, "thermo-chemical."

Abandoning pressure as a cause of the liberation of methane from organic matter does not invalidate Johnson's main conclusion that the formation of methane from entombed organic matter probably increases with depth, because a downward increase would result from the increase of temperature with depth, and also from the greater age of the lower strata, in which the process has been longer in operation.

Naturally I agree with Johnson that the formation of methane crowds away an equivalent volume of water, which must move upward toward the surface because there is no other direction in which escape is possible. The water squeezed out by the settling, consolidation, and cementation of the sediments also must escape upward. Locally also there is reason to believe in the ascent of abyssal fluids, and the pressure of the latter probably supplements that arising from the formation of methane and other gases. I had independently reached the conclusion that there is a general ascent of rock fluids across the strata, from a study of the general distribution of gas pressures, and from the abnormally high chlorine content of oil-field waters.

The evidence of this ascent and the reasons for believing that it is a process in general operation cast doubt on the existence of true connate water in any stratum. The connate water would be shoved out or modified by the entry of other water from below, and rarely could any water strictly be connate to the strata in which it is found. Originally it may have been connate with some lower strata, which in the Appalachian fields would be in early or pre-Paleozoic formations, which were laid down at a time when the sea was comparatively young, and when theoretically the sea-water ratio of Cl:Na may have been as high as that in the waters of deep wells. To some geologists this explanation will be more acceptable than the alternative, that juvenile water rich in calcium and magnesium chloride has entered generally into the strata of oil fields and in many other places; but whatever explanation of this is preferred, the conclusion remains inevitable from any point of view, that water and probably other rock fluids have risen far across the strata. Probably there is no free water left in the older strata that strictly is connate with these formations.

¹ *Journal of the Society of Chemical Industry*, vol. xxii, No. 22, p. 1230 (Nov. 30, 1903).

In the paper on the capillary concentration of gas and oil, I avoided reference to the possible influence of viscosity in promoting the concentration of oil in the larger pores, because it would operate against the apparently identical segregation of gas. Moreover, I have come to think that most of the migration of oil in fine rock pores takes place when the oil is still comparatively light. (I hope soon to present the geological and chemical reasons for believing that all crude oils are descended from lighter parent oils.) At the temperatures prevailing at depths of 2,000 m. light oils have approximately the same viscosity as water, some even less, so that viscosity cannot be an important factor in the concentration or segregation of such oil in its original migration. In the later and comparatively shallow migrations of the condensed heavier oils, such as a typical fuel oil of California, viscosity would be a determining factor in causing the oil from fissures to enter sandstone rather than shale. In fact, such oil could not enter even dry shale, and no oil could enter wet shale from fissures if the conclusions from capillarity are correct. However, fissure migration is a secondary feature, although of great local importance in the formation of some "pools." The original concentration of the oil in sands and fissures must have been due to capillary action operating on comparatively light oils. There is no other way by which the oil of the California fields could have been removed so nearly completely from the diatomaceous and foraminiferal shales in which it is thought to have originated. If the oil were then in its present viscous, highly asphaltic condition, only a very small part of the oil could have been removed and the shales would be generally petroliferous and dark colored, instead of being very locally saturated and generally light colored, as they actually are. In other words, I accept Arnold's conclusion that the oil of California gets heavier as it migrates, and I believe also during its storage in the sands, which is in accord with all sound observation and theory, and I oppose the view of Höfer that crude oil grows lighter with time.

Johnson's three final conclusions seem more than the evidence justifies. We know only that the deep sands of the Appalachian and a few other fields generally are dry. We do not know that the shales are dry. In fact, I suspect that the shales are moist, since the sands still contain water in places. Hence it does not follow that the very deep oil of such fields is scattered in the shale, largely beyond recovery. The value of the very deep reserves is diminished, *first*, by the expense of deep drilling, and *secondly*, by the lack of sufficient water in the sands to control the accumulation of the oil in definite structures. The few definitely synclinal fields, all in dry sands, as near Bluff, Utah, also in southwestern Wyoming and southern Kentucky, have not been very successful. To produce a profitable accumulation of oil in a continuous sand, the sand must contain enough water to greatly restrict the oil-bearing area.

Deep drilling for oil admittedly is a failure in the Appalachian (or

Eastern) fields of the United States; but in California, where the deep sands contain much more water, there is good hope of developing very deep fields. Even now there are a few California wells producing oil from below 4,000 ft.

The two remaining conclusions of the author rest on similar evidence and are subject to the same criticism. The general existence of large deep reserves of natural gas cannot be predicted from the available evidence. Possibly the deep gas may be badly contaminated with abyssal nitrogen, as appears to be the case in the western fields of Kansas. The peridotite dikes of western and of central Kentucky, and the more widely distributed barite-fluorite deposits of the same regions, indicate that abyssal emanations may be important in the deep strata of the sub-Appalachian oil and gas belts. These are interesting possibilities. The evidence to be obtained in deep wells offers great promise of contributions to geological knowledge, in fundamental problems that have received little attention. However, there is little justification for general economic predictions concerning the zone immediately beneath that now explored by the drill.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, Sept., 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Rotary Kilns for Desulphurization and Agglomeration

BY SAMUEL E. DOAK, E. M., PHILADELPHIA, PA.

(San Francisco Meeting, September, 1915)

THE utilization of rotary kilns, of the well-known cement type, for the preparation of iron ores for the blast furnace, has become of considerable economic importance within the past 10 years in certain localities, notably those where the native ores are lean. Such kilns may be used simply for the agglomeration of ores too fine to be charged into the blast furnace (*i.e.*, ores whose loss in the dust catchers is excessive, and whose fineness is detrimental to the working of the furnace), or for the desulphurization of lump and fine ores the sulphur content of which does not exceed 6 or 7 per cent. The desulphurizing action can be combined with that of agglomeration provided the kiln is sufficiently large. Other processes have been developed for the same purpose, but while I am not able to draw a comparison between them by reason of lack of sufficient data, it is safe to say that the kiln, if properly dimensioned for the character of ore to be treated, will compare favorably in its results, both as to economy and quality of product, with any other process. It is not, however, as well adapted for the treatment of flue dust as is a sintering operation, on account of the large percentage of fine coke contained therein. Any combustible entering the kiln at its feed end renders the process very much more expensive and unsatisfactory on account of the difficulty of controlling the temperature, also because of the rapidity with which long "rings" form, due to the increased heat at the cooler end of the kiln.

The greatest value of the kiln at present is in the treatment of pyrites cinder; that is, the residue from pyrites ore after the extraction of the greater part of the sulphur. This cinder, while high in iron content, has too much sulphur remaining in it to be allowed to form a large part of the blast-furnace burden, without further treatment. It will range in metallic iron from 42 per cent., representing a very low-grade cinder from a domestic pyrites, to 64 per cent. of the residue from the best Spanish ore, and the corresponding silica content will vary from the maximum of about 25 per cent. to $1\frac{1}{2}$ per cent. The residual sulphur depends largely on the type of burner used in the original desulphurization. In general, the leanest original ores retain the greatest amounts of sulphur. In

modern practice the percentage of sulphur in the cinder is rarely as great as six, and seldom lower than one and one-half.

Average analyses for iron, silica, and sulphur of the cinders that have come under my personal observation are as follows:

	Iron, Per Cent.	Silica, Per Cent.	Sulphur, Per Cent.
Virginia fines.....	48	18	4.2
Virginia lump.....	54	12	5.0
Spanish fines.....	58	8	2.5
Spanish lump.....	63	2	2.0

In addition, nearly all cinders contain zinc, copper (frequently in sufficient amounts to be extracted commercially as a by-product), calcium, magnesium, etc., and sometimes lead. The total of these accompanying elements, however, is usually well under 6 per cent.

The process is a simple roasting, with the addition in the case of fine ores of a nodulizing or agglomerating operation, performed near the discharge end of the kiln, at a higher temperature than is necessary for the desulphurization. The sulphur is combined in the ore as sulphide and sulphate of the various bases, and if the kiln is not forced over its capacity there is no difficulty in reducing it to less than 0.05 per cent. of the finished product. As far as the desulphurizing action is concerned, the principal caution to be observed is not to allow the kiln to get too hot at the back end. To get a dead roast the operation should be very gradual, and the heat should range from a barely perceptible dull red at the feed end to a bright yellow at the discharge end. When the ore is too hot in the center of the kiln, there seems to be a tendency to hinder the desulphurization, probably on account of an almost imperceptible glazing of the particles of cinder, that renders them impervious. For this reason it is important that the roasting should be complete before the nodulizing begins, since for the success of the latter operation the ore should be sufficiently pasty to ball up. This pastiness, however, should never be carried to the point where there is any flow of the cinder, and in fact it should never be visible in the kiln. A bright yellow heat is sufficient for the nodulizing, and it is well to keep the zone of agglomeration as near the discharge end of the kiln as possible, 15 ft. being sufficient length for the operation in any kiln not more than 100 ft. in length. For this reason a short flame is very desirable.

Oxygen necessary for the desulphurization is furnished either by stack draft at the feed end, by a blower at the discharge end (in connection with the fuel supply), or by both. The height of the stack is determined more by the necessity of raising the sulphur fumes above the height where they are obnoxious to people, and harmful to vegetation, than by the oxygen requirements of the process, as the slope of the kiln together with the draft induced by the coal blower is sufficient in small

kilns to furnish the required amount of air without a stack. A kiln 60 by 5 ft. was run for several months without any stack, with excellent results, the only artificial draft being supplied with the powdered coal by an Aero pulverizer.

This same kiln on an 80 by 4 ft. stack gave incomplete desulphurization and an excessive amount of "ring" trouble, due to too high a temperature at the feed end.

At present the rotary kiln owes what popularity it possesses to its nodulizing feature, in spite of the fact that it is a cheap and efficient roaster, and that it is much more successful, as a roaster, working on coarse than on fine ore. As I remarked above, the sulphur should be practically eliminated from the ore before nodulizing begins. The most satisfactory way of insuring this would be to separate the two processes by using two kilns, but that would be impracticable except in a very large plant, and as a matter of fact the whole operation can be done very well in one kiln provided it is not too small. The greatest advantage of separating the processes would be that "ringing" could be almost entirely eliminated in the desulphurizing kilns, and confined to the nodulizing kilns, which could be made short and of large diameter and great capacity.

It has been the custom arbitrarily to connect a certain diameter with a certain length of kiln. A 60-ft. kiln is usually 5 ft. in diameter; 80- and 85-ft. kilns are 6 or 7 ft. in diameter, and a 100-ft. kiln is ordinarily 8 ft. in diameter, but it would frequently be more economical to proportion the kiln in a less haphazard way than by accepting the builder's standard designs. The two determining factors should be the output desired, and the quality of ore to be roasted. The output is dependent on the dimensions of a cross-section of the bed of ore that can be carried in the kiln, the slope of the kiln, and its speed of revolution. The slope of the kiln, however, can be left out of consideration. Good practice places it at $\frac{3}{4}$ in. to the foot, and increased or decreased output can be better obtained by changing either of the other variables than by changing this. Speed control, too, is a feature of the design of every kiln now, so that this factor may be adjusted to the most satisfactory point after the kiln is in operation. The question of output, then, so far as the kiln design is concerned, is determined by the width and depth of the ore bed. In general, the latter can be increased with the length of the kiln, since the length of time it takes the ore to pass through the kiln will be greater as the kiln grows longer, all other things being equal, but the increased capacity due to the length will not be very great, since, without changing the diameter, the ore bed cannot be deepened to any great extent without harming the roasting process, for which a continual and complete stirring and overturning of the ore bed is necessary. Too great a depth of bed prevents this.

An increase in the diameter of the kiln, on the other hand, increases

the possibilities of the output considerably, by allowing the cross-section to be made much greater with a slight increase in depth, giving the desired effect of a greater ratio of surface of ore exposed to the air in comparison with the area of the cross-section. It would seem reasonable, then, to base the length of the kiln entirely upon the time necessary for desulphurization and agglomeration, and to control the output by the diameter.

The question of the speed of progression of the ore in the kiln, upon which, together with the kiln length, depends the time available for treatment, cannot be accurately determined beforehand, as it is controlled by too many variables. The size of the ore particles, and their shape, the depth of bed, condition of the lining, and the size and shape of the ring all have an effect on the speed of progression of the charge. Fortunately, however, we know from actual practice that any type of cinder can be treated satisfactorily in kilns 80 to 100 ft. in length, turning at a speed of from one revolution in 60 sec. to one in 90 sec. The greater length is safer in treating all fine, high-sulphur cinders, since it reduces greatly the chances of a poor product through careless supervision.

The tonnage output of a kiln varies greatly with the kind of cinder charged. A kiln 60 by 5 ft. with $4\frac{1}{2}$ -in. lining will treat 1,500 tons a month of low-sulphur lump ore, while on high-sulphur fines, it will with difficulty take care of 750 tons, and frequently even at this low tonnage a large percentage of the product will not be nodulized. A kiln 85 by 7 ft. will handle 3,000 tons of high-grade cinder per month, and about 1,500 tons of the poorest grades, while a kiln 100 by 8 ft. will vary from 4,500 to 2,500 tons with the quality of cinder treated. A kiln 85 by 8 ft. should handle almost as much as the one 100 ft. long, but at a greater fuel cost. In general, the larger kiln will have the lower fuel cost and the fine domestic ores will need less coal, on account of their higher sulphur content. On a 60 by 5 ft. kiln, the fuel consumption averaged for several months about 12 per cent. by weight of the weight of roasted ore.

The mechanical difficulties that were so common when rotary kilns were first used for the treatment of iron ores have been practically eliminated. The most important things to be observed to insure continuous operation are, to make the kiln sufficiently strong (reinforcing it especially very heavily under the tires), and to keep the operating machinery well protected from the dust. The tires should always be kept tight to the shell, as if any play is allowed the bolt holes will gradually become enlarged, and the kiln will bear down hill so strongly that it will frequently break its thrust bearings. At least an inch of additional metal should be put around the shell under the tire.

The one great obstacle now in the way of continuous operation is the formation of "rings," caused by the cinder becoming hot enough to stick to the lining. As soon as the lining is covered the "ring" increases more rapidly by building up upon itself around an ever-decreasing circum-

ference until it seriously interferes with the operation, and has to be removed. It usually forms in the 15 or 20 ft. nearest the discharge end of the kiln, and puts an excessive amount of weight upon that end. A piece of the "ring" is hard and has the appearance of having been partly fused. It looks very much like mill cinder.

Many devices have been tried to prevent the formation of the "ring," or to remove it without stopping the kiln, but, so far as I know, none of them have been successful. The best plan, in my experience, has been to shut down the kiln, when the ring became too large, cool it off with water, and cut it out with bars and picks. It is impossible, I feel sure, to prevent its formation when running on fine ores. With low-sulphur lump ores that do not need to be nodulized, careful control of the heat will allow a run of indefinite length, but as all the lump cinders (unless screened) carry a considerable percentage of fines (*i.e.*, -20 mesh), sooner or later a shutdown will be necessary. On any kind of cinder, however, the formation of the ring can be postponed by keeping, as nearly as possible, the conditions of temperature, speed, and feed, from fluctuating. The temperature should always be kept as low as is compatible with the successful operation of the kiln.

A 9-in. firebrick is generally used for lining, and is probably the best size on the larger kilns. A $4\frac{1}{2}$ -in. brick is satisfactory for kilns of 5 and 6 ft. diameter. The thicker bricks protect the shell better and prevent warping, but in the smaller kilns they reduce the available diameter too much. The lining wears down rapidly, and frequently individual bricks are loosened or broken in cutting out the ring. If the shell becomes warped under the tires, as is frequently the case when the kiln is not reinforced strongly enough at this point, it is very difficult to keep the bricks in. A $9\frac{1}{2}$ -in. lining should last over a year in the hot part of the kiln where the ring forms, and very much longer in the cooler end.

Automatic feeders are frequently used to insure a constant rate of charging, but they are really unnecessary, as the slow motion of the kiln will effect this, if only a simple feed pipe is used. All machinery used in handling the ore and the fuel and in rotating the kiln should be as nearly dust proof as possible, and should be driven by fully inclosed motors, as the fine cinder dust is very harmful. The best plan is to keep the feeding apparatus, the machinery for handling the roasted ore, and the fuel pulverizer and blower, all under separate housings. The kiln itself does not need to be housed, except where such housing is necessary to protect its driving machinery. Powdered coal is used as the fuel, so far as I know, exclusively, with the exception of one or two plants where blast-furnace gas has been tried. An oil flame would probably be very satisfactory where cheap oil could be obtained, on account of the ease of controlling the temperature incident to its use.

The cost of production of a ton of roasted ore, aside from the cost of

the raw cinder, depends chiefly on the labor cost. Two men on a shift should be able to take care of a well-designed plant producing up to 100 tons in 24 hr., while in a poorly designed plant this number might easily reach four or five. On a small kiln, 60 by 5 ft., the labor cost per ton, on high-sulphur fine cinder, averaged 40c. for several months, and with the same kiln, using high-grade lump cinder, this figure was reduced to 30c., but both these figures were excessive on account of faulty design, that made necessary three and four men on a shift; also on account of a very low output per man, due to the small size of the kiln. On the other hand, this was partly balanced by the fact that wages were low, and only two shifts were necessary. Repairs and supplies should not cost over 8c. per ton of roasted ore on a kiln of any size. The important factor in economic kiln operation is careful designing, and it cannot be said too strongly that the cost of operation of a rotary kiln will be almost entirely determined by the layout of the plant.

The roasted ore, for which the cement term of clinker is used, works very well in the furnace. In Virginia practice, where the blast pressure is comparatively low, it is not essential that the nodules should be large. If 80 per cent. of the clinker stays on a screen of $\frac{1}{8}$ -in. mesh, there will be no difficulty in the furnace at pressures of under 9 or 10 lb. It is very easy to make larger nodules, but as a higher temperature is required to do so, with the attendant evils of more frequent "ringing," greater fuel expense, and a harder, less porous product, it is better to keep the nodules as small as possible. The clinker frequently forms from one-sixteenth to one-half of the burden without ill effects, working with "brown ore" and specular hematites, and by its use the yield of the Southern furnaces where the native ores are lean can be raised from a scant 42 per cent. to 50 per cent. and over, depending on the quality of the cinder. It is in localities where the native ore is lean that the kiln is valuable. It affords an opportunity of enriching the burden with a high-grade ore at a much lower cost than could be done by direct purchase of lake or other rich ores, and where the silica in the native ore averages well over 20 per cent., it is indispensable that the burden should be sweetened in some way to cut down the excessive amount of flux and consequent fuel necessary to produce good iron.

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The Mexican Oil Fields

BY L. G. HUNTLEY, * PITTSBURGH, PA.

(San Francisco Meeting, September, 1915)

I. HISTORY OF OIL DEVELOPMENT IN MEXICO

THE occurrence of oil or "tar" in Mexico was mentioned as early as the seventeenth century by Friar Sagahun, who gives the Indian name "chapopote," by which these asphalt seepages are still called. This asphalt was apparently used, as it has been used by primitive people in many parts of the world, in religious ceremonies and for medicinal purposes. Travellers also report that the ruins in Yucatan and the pyramids in southern Mexico show traces of the use of "chapopote" as a building cement.¹

DeGolyer says that the first attempt to exploit oil or gas in Mexico in a commercial way is shown by the records of the Memoria de Fomento of 1865, when permission was granted to a Senor Ildefonso Lopez to exploit the deposits of petroliferous substances in the San Jose de las Rusias area in the State of Tamaulipas. Other concessions follow, and probably refer to surface seepages which occur in that district. This was several years after the discovery of oil in Pennsylvania. In 1868, a well 125 ft. deep was drilled by a company organized in Mexico City, in what is now known as the Furbero district, and a little oil was refined there.

In 1873, residents of Tampico denounced seepages along the Tamesi River, and asphalt was mined near Tempoal in the Canton of Tantoyuca. No drilling was attempted. But between 1880 and 1883 several shallow wells were drilled for oil in Mexico, two of them being near the present Potrero de Llano field west of Tuxpam. The wells were drilled by a Boston company with Canadian rigs, and are said to have reached a depth of about 400 ft., while one is reported to have flowed. Several other abortive attempts were made to drill for oil between 1885 and the beginning of the present century, about which time the Mexican Petroleum & Liquid Fuel Co., Ltd., in which Cecil Rhodes was interested drilled 24 wells unsuccessfully, several of them as deep as 1,500 ft., in the State

* Johnson and Huntley, Consulting Oil Geologists.

¹ De Golyer. *Oil and Gas Journal*, April 16, 1914.

of Vera Cruz. Also Messrs. Doheny and Canfield of California purchased land in the State of San Luis Potosi, and in May, 1901, the first successful well was struck in the present field of Ebano, 50 miles west of Tampico. Various shallow wells were drilled about this time in the States of Vera Cruz, Tamaulipas, near Guadalupe, Chiapas, and Tabasco on the Isthmus of Tehauntepec; and several asphalt companies attempted to mine the surface deposits in the Cantons of Tuxpam and Ozuluama.

In 1904, drilling was commenced by the Oil Fields of Mexico Co. (Pearson interest) at Furbero, and its first well was brought in successfully. Pez No. 1, the first large well on the Ebano property, was also brought in during the same year, and has flowed continuously since. About 1907, drilling became more active, especially in the Cantons of Tuxpam and Ozuluama. It was started at San Diego by the Pennsylvania Oil Co., at Tumbadero by the Pearson interests, and at Juan Casiano and Tres Hermanos (La Pithaya) by the Huasteca Petroleum Co. A Mexican bank drilled four shallow wells near the Tampalache seepage, just north of the Panuco field. Operations were also commenced by Pittsburgh interests at Los Esteros, in the State of Tamaulipas.

In 1908, the Pearson refinery at Minatitlan on the Isthmus of Tehauntepec commenced operations. In May of the same year San Diego No. 2, the discovery well of the Dos Bocas field, came in with a production of 2,500 bbl. daily. On July 4, the famous San Diego No. 3 was brought in, caught fire immediately, and burned until extinguished by encroaching salt water. The Chijol field in the Ebano district and the Topila field were brought in during this year, which inaugurated important operations in the Mexican oil fields.

The year 1910 saw the discovery well of the Panuco field (East Coast Oil Co. No. 401) drilled, and the first well in the Tanguijo and San Pedro districts were put down in that year by the Pearson interest, the Mexican Eagle Petroleum Co. The Huasteca Petroleum Co. also brought in its No. 6 and No. 7 Juan Casiano, one with 14,000 bbl. and the other with 28,000 bbl. daily production. About the time these came in, this company completed its 8-in. pipe line from that field to Tampico. The Mexican Eagle Co., having completed its pipe line from Furbero to Tuxpam, brought in, in December, 1910, the famous Potrero del Llano well, yielding about 100,000 bbl. daily. Drilling was by this time very active; and in 1911 the first exportations of crude oil were made to the United States from Tampico.

In July, 1912, the Government put a tax of 20 centavos per metric ton (1.1 c. U. S. gold per barrel) upon all oil produced in the Republic. The construction of a large fleet of tank steamers had already been commenced by the Huasteca Petroleum Co. In 1913, the Penn-Mex Fuel Co., now a subsidiary of the Standard Oil Company of New Jersey, through the

South Penn Oil Co., brought in two good wells in the Alamo field near Potrero del Llano. The Chila-Salinas and the Topila field began to suffer severely from the invasion of salt water. The Mexican Eagle Co. brought in the discovery well in the Los Naranjos field in Amatlan, which has since broken its valve and proved to be a much larger producer than was estimated by flowing at the rate of 50,000 bbl. per day for a week. On December 1, the Government raised the production-tax to 75 centavos per ton (approximately 4c. U. S. gold per barrel) besides increasing numerous duties affecting the petroleum industry. A bill was proposed in the Chamber of Deputies for the nationalization of the petroleum industry, but no decisive action was taken.

During January, 1914, the Dutch Shell interest (La Corona Petroleum Company) brought in its big well, which is estimated to have a capacity of 100,000 bbl. per day, in the Panuco field. During the spring of this year the activity of both the Federals and the Constitutionals became very great in the vicinity of Tampico, culminating in a general exodus of the foreign employees of the oil companies in April, at the time of the Vera Cruz affair. Operations came to a standstill for a time, but the foreign employees gradually returned during the next two months, and restricted drilling was carried on by some of the larger companies, especially in the Panuco field. But continued guerrilla warfare, and the generally hostile attitude of all Mexican factions, led to heavy demands upon all companies, so that affairs in the oil fields were by no means normal when the European war broke out in August. In addition to this, lightning set fire to the big Potrero del Llano No. 4 well, which was not entirely extinguished until after the end of the year. During the latter part of the year hardly a string of tools was running. Gen. Carranza meanwhile appointed a commission to examine into the oil industry in Mexico and to make recommendations as to future methods of regulation. The conditions under which new work might be carried on under this decree were prohibitory, and resulted in a complete shutting down of all work during the early part of 1915. However, the Carranza government later modified its attitude, and agreed to issue permits to companies making application, under which they could operate under certain restrictions, pending the result of the work of the commission. This remains the situation in June, 1915, and development work is being carried on under difficulties, with only about six rigs running. Oil shipments have not been interrupted, however.

II. PRODUCTION

The following table shows the annual production of petroleum in the Mexican fields since 1904. Within the last three years, this has not represented the productive capacity, which was much greater. The completion of additional transportation lines to seaboard, the building of more tank ships, and the lessening of the over-production of high-grade

oil in the United States, together with the broadening market following the end of the European trouble, will see the production of the Mexican fields advance enormously. At present they are yielding only about 70,000 bbl. per day; but the wells already drilled could produce 330,000 bbl. per day. In fact, the latest figures on good authority make this capacity more than 500,000 bbl.

PRODUCTION OF OIL IN MEXICO

Year	Barrels (42 gal.)	Year	Barrels (42 gal.)
1904	200,000	1910	4,099,000
1905	300,000	1911	13,655,488
1906	500,000	1912	16,844,066
1907	1,000,000	1913	24,574,500
1908	3,481,000	1914	25,725,403
1909	2,765,000		

During the latter part of 1914 there were 56 companies actually operating in the Mexican fields. But some 200 companies had been organized and were engaged in the business in one way or another. Most of these were American companies, the exceptions being the Pearson interests (English), the Dutch Shell interest (English and Dutch), and a number of Mexican companies particularly in the Panuco and Topila field. The principal shippers of petroleum were the Mexican Eagle Oil Co., Ltd. (Pearson); the Huasteca Petroleum Co. (Doheny); the Tampico Company; the Mexican Gulf Oil Co. (Mellon); the East Coast Oil Company (Southern Pacific Ry.); the Trans-continental Petroleum Co. (John Hays Hammond); La Corona Petroleum Co. (Dutch-Shell); the Panuco Valley Oilfields Corp., Ltd. (Simms & Bowser); and the Penn-Mex Fuel Co. (Standard Oil Company of New Jersey).

Not more than 50 wells were actually producing, and more than half of the oil produced came from about six of these. The rest were shut in, waiting a better market, or better transportation, or for other reasons more or less connected with business policy, which sometimes included the obtaining of additional territory or the clearing up of titles.

The estimated total daily capacity of all wells in the Mexican fields of 330,000 bbl. may be divided among producing fields as follows, at the beginning of the present year:

Barrels	
150,000 Panuco	} Northern district, producing heavy oil, from 10 to 14° Bé.
4,000 Topila	
6,000 Ebano and Chijol	
65,000 Casiano and Los Naranjos	} Southern district, producing oil from 18 to 27° Bé.
100,000 Potrero, Alazan, Alamo, Tanguijo, and other points shipping from Tuxpam	

NOTE.—The Huasteca Petroleum Company's production at Cerro Azul and Juan Felipe has not been included through lack of information; but it is claimed that these wells are capable of adding 175,000 bbl. a day to this estimate, making the total 500,000 bbl.

While these figures indicate what might be expected of the Mexican fields if transportation and marketing facilities were to make it possible for all wells to be drawn upon to their full capacity, yet as a matter of fact, the year 1914 saw only an average of 70,000 bbl. per day of actual production distributed as follows:

	Barrels
Topila.....	398,679
Southern fields.....	18,830,359
Panuco.....	5,058,970
Chila-Salinas, Ebano, Chijol, etc.....	1,108,995
Tehuantepec.....	328,500
Total Production.....	25,725,403

This was consumed or stored as follows:

	Barrels
Clearances.....	20,674,357
Internal consumption.....	1,172,898
Refinery products.....	2,000,000
To storage.....	1,888,148

The addition to stocks brought the total storage in Mexico at the beginning of the year to 15,127,834 bbl. as follows: In steel storage, 9,658,258 bbl.; in earthen storage 5,369,576 bbl. (including 508,690 bbl. unmerchable).

Clearances from Mexican ports were distributed as follows during 1914:

	Barrels
To the United States.....	15,476,727
Mexican Coastwise.....	4,510,061
South America.....	195,138
To the Continent (Europe).....	356,205
To the United Kingdom.....	69,780
To Panama.....	46,446
To Cuba.....	20,000
	20,674,357

It is not to be expected that most of the producing districts in the Mexican fields will decline in a manner similar to the sand pools of the United States. It has been proved that the estimated initial capacity in many wells would be equaled if not surpassed by actual performance if they were allowed to flow freely. In fact, wells such as the Juan Casiano No. 6 and No. 7 and Los Naranjos No. 1 and No. 4, and the Mexican Fuel Oil Company's No. 3 Zurita, all in different districts, increased their production after being allowed to flow, as shown below:

	Initial Production, Barrels	Actual Flow Later, Barrels
Juan Casiano, No. 6.....	10,000	14,000
Juan Casiano, No. 7.....	20,000	28,000
Los Naranjos, No. 1.....	5,000	10,000
Los Naranjos, No. 4.....	10,000	50,000
Zurita, No. 3 (Panuco).....	30,000	80,000

These cannot be considered exceptional cases, since so few of the large wells have ever been allowed to flow freely. Considering geological conditions, which will be discussed later, such behavior is to be expected. The oldest commercial wells in the Mexican fields (Pez No. 1 in the Ebano district, and Juan Casiano No. 7) after producing since 1904 and 1910 respectively, and from widely separated districts have shown practically no decrease in their flow since they were drilled in. The famous Potrero del Llano well, in the southern part of the field, has never produced up to its capacity and is still yielding as much as ever. These last two wells have produced approximately 40,000,000 bbl. of oil each during their history, and show practically no signs of decline, so far as known. The same may be said of the oldest well in the Panuco field (East Coast No. 401), drilled in 1910. This is even more remarkable, since the area surrounding this well has been closely drilled. While a few of the smaller wells at Panuco showed a decline after being shut in for a time, yet this apparently did not indicate any exhaustion, for at the same time No. 3 Zurita increased its production, as indicated above.

It must be said that producing conditions in Mexico differ fundamentally from the fields in the United States, and that, by reason of these conditions there is no vital need on the part of producing companies to bring the oil to the surface as quickly as possible (as at Cushing), and hence relatively little necessity for large tankage—except at terminal stations for operating reasons. The Mexican oil production comes from about 20 different distinct groups of wells. In all except Panuco and Topila, one company usually controls a large acreage surrounding its group of wells. Hence, each of these 20 districts represents a pool, or geological unit, in which only a few initial wells have been drilled. It is thus to the interest of each company to drill efficiently, and to keep wells shut in when there is no market or no transportation. There is usually no waste of oil or gas, except in the case of accidents, or inability to control a well. One of the few cases of waste happened recently, where a company struck a gas well south of the Topila field, which it is allowing to blow wide open, hoping it will show oil. This is almost the ideal condition advocated by some conservationists in this country—one company to each pool.

There has been very little decline of pressure as yet, even in the older pools; and wells such as the first ones at Ebano and Juan Casiano, producing since 1904 and 1910 respectively, are doing so at practically their original rates. In the latter case 40,000,000 bbl. of oil have been taken from the one well, with not more than 10 per cent. decline from its initial daily production. If this locality had been drilled up, as was the case at Cushing, probably it would almost have ceased to produce by this time, and might not have produced as much oil from all wells as will be obtained from the few holes which will be drilled by the present company.

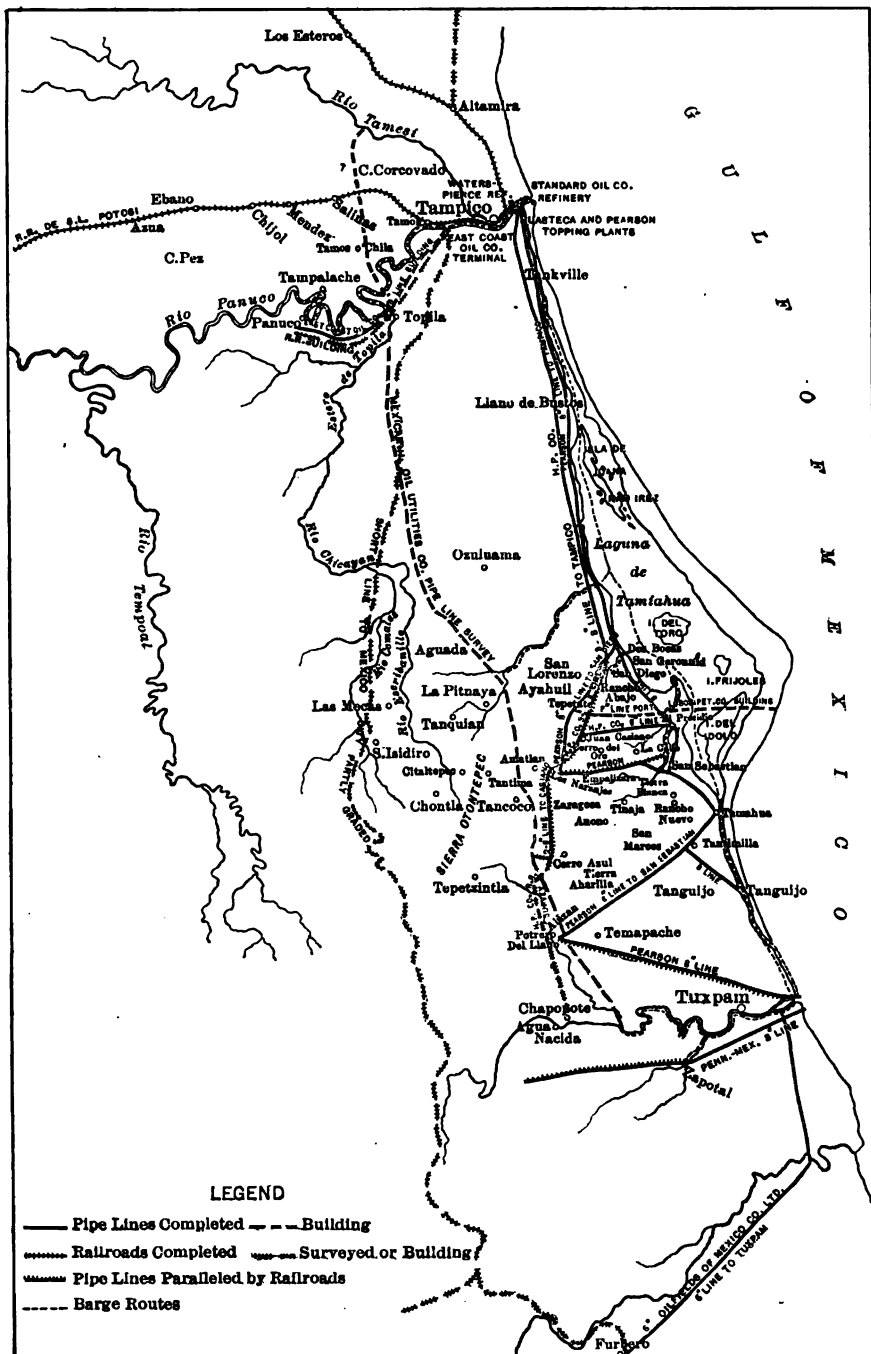


FIG. 1.—SKETCH MAP OF THE MEXICAN OIL FIELDS, SHOWING PIPE LINES AND RAILROADS.

Transportation.—Oil from the Ebano district is used largely as fuel by the Mexican railways, and considerable asphalt for paving also goes by rail. The Panuco oil all goes in barges to terminals near Tampico, and thence in tank ships to destination. The other fields all pipe their oil to seaboard either at Tuxpam or Tampico, where it is taken by tankers. There are at present 427 miles of pipe line in Mexico, mostly 8 in., some is 6 in. and smaller. About 50 miles additional are building or contemplated. The principal lines are as follows (see Fig. 1):

East Coast Oil Co.; 8-in. line, Panuco-Ponce; about 13.65 miles completed of a 24-mile line to Tampico.

Mexican Eagle Co. (Pearson); 8-in. line, Potrero del Llano to Tuxpam; 8-in. line, Potrero del Llano to Tierra Amarilla, to Tanguijo, to Tamiahua, to San Sebastian, to San Diego, to Bustos, to Tampico; 8-in. Los Naranjos to San Diego. (An allied company has a 6-in. line from Furbero to Tuxpam.)

Huasteca Petroleum Co. (Doheny); two 8-in. lines from Cerro Azul to Juan Casiano; three 8-in. lines from Juan Casiano to San Geronimo, to Esperanza, to Tankville, to Tampico; and a gas line from Juan Casiano to Tampico.

Penn-Mex Fuel Co. (South Penn Oil Co.); 8-in. line from Alamo and Agua Nacida to Tuxpam; 4-in. water line paralleling above.

Transportation is a serious difficulty in getting Mexican oil to the market. Present equipment could not possibly handle the full production of all wells hitherto drilled, which is variously estimated from 330,000 to 500,000 bbl. per day. In 1913, the production of these wells was 24,574,500 bbl. During 1914 none of the old wells declined in production, and 32 new wells came in, representing a possible new production of 196,000 bbl. per day. During the same year a number of pipe lines were enlarged from 6 in. to 8 in., and at least one new field (Los Naranjos) was connected with the seaboard. Shipping facilities from the Panuco field were increased, and some new tankers went into service. In spite of all this new production and new equipment, the field yielded only 25,725,403 bbl. of oil during 1914. At the present time not more than 50 miles of additional pipe line are under construction or contemplated, and very few new tankers, considering this great shut-in production. This indicates that companies are awaiting a better market before making a serious effort to get their product from wells already drilled to points of consumption. A field with a possible daily production of at least 330,000 bbl., which meanwhile only produces 70,000 bbl. per day, and at the same time makes little effort to build new pipe lines and tankers, as indicated by present constructions under way, furnishes strong evidence of the effect of market conditions.

---It is to be assumed that development in Mexico will be relatively much slower than would be the case if these fields were located within the United States, for the following reasons:

1. The great expense of operating keeps out all except financially strong companies which will not develop their territory so rapidly as to break the market, and, since they control large areas around their wells can afford to suspend production, awaiting a better market, because they are not losing pressure or oil to their neighbors in the meantime.

2. Political difficulties retard development.

3. Government and State taxes, high-operating and maintenance charges, as well as the cost of transportation by tank ships, add to the cost of the oil delivered to markets such as the United States. These expenses make it impossible to put oil into United States markets for less than 35 c. per barrel at Gulf Coast ports or approximately 50 c. for Atlantic ports. The price for this oil cannot break below these costs for any except small lots, offered at times of a crisis in the field. This is particularly true, since the largest producers also control, either in Mexico or in the United States, their own pipe lines, tankers and refineries.

It may be added that Mexican oil at those prices, plus a profit, must compete with Gulf Coast and California fuel oil, and Oklahoma residues. The fact that it cannot be brought to this country and sold at a price much below these domestic oils prevents its being diverted into as many new channels as usually open when the price breaks to a low point; hence the market will not expand so rapidly. In so far as transportation is a factor of this relatively high price for Mexican oil, it is the reason for this restricted market. And it must be admitted that transportation is thus indirectly the chief factor in keeping down production.

III. OPERATIONS IN DETAIL

The following was the approximate status of all operations in the Mexican fields during the spring of 1915:

MEXICAN EAGLE PETROLEUM COMPANY, LTD. (Pearson interest)

Well No.	Property	Daily Production Barrels	Remarks
1, 2, 3 & 4	Dos Bocas (Lease) Camp abandoned	Small 200,000	Nos. 1, 2, & 3 were small producers. No. 4 came in at 200,000 bbl. a day and burned for 57 days, ruining the field
1, 2 & 3	San Antonio de Tamihui (Lease)	Dry holes, all more than 3,000 ft. deep	
	Potrero del Llano (Lease)		Wells about 2,000 ft. deep
1		800	Shut in
2		600	Shut in
3		Dry hole	
4		110,000	Gusher
5			Shut down
6			Abandoned
7 & 8			Drilling
	Alazan Ranch (Lease)		Adjoining Potrero del Llano
1 & 5			Abandoned
2 & 3			Salt water
4		11,000	Shut in
6		3,000	Shut in
7		2,000	Shut in
8			Crooked hole
9		3,000	Shut in
10		1,000	Shut in
11, 12 & 13			Drilling
	Tierra Amarilla (Lease)		South Field
1 to 5			Salt water and small amount of oil
6		Can produce 6,000 bbl. oil and 4,000 salt water	Shut in
7, 8, 9, & 10			Producing salt water and small amount of oil.
11 to 15			Drilling or derricks
16			Location

MEXICAN EAGLE PETROLEUM COMPANY, LTD. (Pearson interest).—Continued

Well No.	Property	Daily Production Barrels	Remarks
.....	Tanguijo (Lease)		South field
1 to 20.....			Total production about 500 bbl. a day. Nos. 2, 5, 6, 7, 8, 9, 10, 11, 13, 15, 17, and 18 are shallow wells about 1,300 ft. deep. No. 20, 1,750 ft., 20 gravity oil, came in Mar. 11, 1914, located 3 miles from Nos. 1 to 19 inc.
.....	Naranjos (Lease and fee)		Amatlan 20 gravity oil
1.....		10,000	1,885 ft. deep
2 & 3.....			Dry holes, about 2,300 ft.
4.....		50,000	Flowed this amount for a week when gate valve broke
5, 6, & 7.....			Locations
.....	San Marcos (Lease)		South field, 20 gravity
1 & 3.....			Dry at about 2,800 ft.
2.....		150	Shut in
.....	Tierra Blanca (Lease)		35 gravity oil
1.....			Abandoned
2.....		20	Shut in; 3,100 ft. deep
.....	Tlacolula (Lease)		
1 & 2.....			Dry holes at 4,100 ft.
3 & 4.....			Drilling and derrick
.....	San Pedro (Lease)		State San Luis Potosi
1 & 2.....		10 (each)	Shut in; 45 gravity oil
3 & 4.....			Drilling

HUASTECA PETROLEUM COMPANY (Doheny Interest)

.....	Juan Casiano (Fee and lease)		Chinampa, 19 gravity
1.....		400	Shut in; 1,700 ft.
2.....		100	Shut in; 1,700 ft.
3, 4 & 5.....			Abandoned
6.....		10,000	Shut in; 2,070 ft.
7.....		25,000	Has been flowing 5 yr.; 1,700 ft.
8 & 9.....			Drilling

HUASTECA PETROLEUM COMPANY (DOHENY INTEREST).—(Continued)

Well No.	Property	Daily Production	Remarks
.....	Cerro Azul (fee)		South field
.....			Reported 27 gravity oil
1.....			Abandoned
2 to 5 inc.....			Shut in
.....	Juan Felipe (fee)		South field
1.....			Shut in
.....		Five wells are reported completed on these two properties, with a combined capacity of 175,000 bbl. daily. Pressures reported from 575 to 1,080 lb. for different wells	
.....	Pithaya (fee)		Northwest of Casiano
1 & 2.....			Abandoned
3 & 4.....			Drilling
.....	Juarez (sub-lease)		Adjoining Pithaya
1 & 2.....	Show in No. 1		Shut down
.....	Ebano field		Fifty miles west of Tampico
1 to 27.....		8,000	All but 5 shut in; 11 gravity oil

PENN-MEX FUEL COMPANY (SOUTH PENN OIL Co.)

.....	Alamo Ranch (Lease)		South of Tuxpam River
1.....		2,000	
2.....		20,000	
3 & 4.....			Drilling
.....	Agua Nacida (Lease)		South of Tuxpam River
1.....		Show	
2.....			Abandoned 2,700 ft.
3.....			Drilling
4.....			Derrick
.....	Tamatoca (Lease)		Near Potrero del Llano
1.....			Drilling at 2,950 ft. last year
.....	Las Canas (Lease)		South of Tuxpam River
1.....			Drilling at 3,000 ft.
.....	Molino (Lease)		
1.....			Derrick

INTERNATIONAL PETROLEUM COMPANY (JOHN HAYS HAMMOND INTERESTS)

Well No.	Property	Daily Production	Remarks
1 & 2	Tinaja Ranch		South of Amatlan Dry holes
1	San Marcos		Near Tanguijo, shut down at 3,200 ft.
2			Derrick

MEXICAN EASTERN OIL COMPANY

No. 1	San Diego Ranch (Near Dos Bocas)		Abandoned at 2,785 ft.
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ELECTRA OIL COMPANY

1	Rancho Abajo. (East of Casiano)	Show oil	Drilling
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MEXICAN PREMIER OIL COMPANY (SPELLACY)

1	Soledad Ranch. (north of Cuecil- los)		Shut down at 2,350 ft.
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HIDALGO PETROLEUM CO. (M. A. SPELLACY ET AL)

1	La Calle		Rigging up
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CORTEZ OIL COMPANY (AMERICAN TOBACCO CO. INTERESTS)
(Now the Port Lobos Petroleum Co.)

3	Empalizada		Shut down at 2,700 ft.
4	(Amatlan) north of Casiano)	25,000	Shut in. Casing trouble
1	Sandoval		Drilling (offsetting No. 4 Tepetate)

GALEY ET AL

Well No.	Property	Daily Production	Remarks
1.....	San Diego de la Mar.....		In Chinampa Shut down at 2,100 ft.

STANDARD OIL COMPANY OF MEXICO

1.....	El Gallo Ranch (Lease).....		Abandoned at 2,727 ft.
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MEXICAN FUEL OIL COMPANY

1.....	Caracol Ranch (11 to 12 gravity oil).....		West of Tampico
2, 3 & 7.....		350	Abandoned, shallow Shut in, shallow
4, 5 & 9.....			Shut down
8.....		175.	Flowing, 770 ft. deep

TAMPICO FRUIT COMPANY

1.....	Caracol Ranch.....	2	Fifteen miles up the Panuco River Shut in 2,910 ft.
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TAMPICO OIL Co., LTD.

1 & 4.....	Chila Ranch.....	205	Near Caracol; 12 Gravity Flowing
2 & 3.....		50	Shut in
2.....	Salinas Ranch.....	100	16 gravity oil Sanded up; 1,603 ft.
3.....			Drilling

DOS BANDERAS OIL & GAS Co.

1.....	Dos Banderas lease (Near Tampico)	Small pro- duction	Shut down; 3,480 ft. deep
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SMITH OIL Co.

No. 1.....	Caracol lease.....		Drilling at 1,925 ft.
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Topila Field

Company	Wells, Completed	Wells, Drilling and Locations	Wells, Dry or Abandoned	Estimated Daily Pro- duction, Barrels
Mexican Fuel Co. (Pierce Oil Corp.)	4	1	1	750
East Coast Oil Co. (Southern Pacific R.R.).....	3	3	1	100
Mexican Oil Co.....	1	1		2,000
American Fuel Oil Co.....	1			5,000
La Corona (Dutch Shell).....	2	1	1	130
Mexican Gulf Oil Co. (Mellon interests).....	1		1	
Tampico Panuco Oil fields.....	1	2	1	
Topila Petroleum Co. (Spellacy)...	1		1	
Penn-Mex Fuel Co. (South Penn)	1		1	
Scottish Mexican Oil Co.....	1		1	
M. C. Anderson.....	2			52
W. H. Milliken.....	1			20
R. L. Brooks (Producers Oil Co.)	1			450
Cia. Petrolera de Monterey	1	2		105
Mexican Associated Oil Co.....	1		1	
Cia. Explotadora de Topila	1	1		

Panuco Field

Company	Wells, Completed	Wells, Drilling and Locations	Wells, Dry or Abandoned	Estimated Daily Pro- duction, Barrels
Vera Cruz Mexican Oil Syndicate, Ltd.....	5	1		10,000
International Petroleum Co.....	4	4	2	800
Transcontinental Petroleum Co.....	1	1		800
Gulf Coast Corp.....	3	1		100
Mexican Development Co.....	1		1	
La Corona Petroleum Co.....	4	2	1	100,600
Freeport and Mex. Fuel Oil Corpor- ation.....	5	4	1	31,500
National Oil Co.....	9	3	1	21,000
Mexican National Oil Co.....	2	2	1	
Tampascas Oil Co.....	2	1	1	350
Penn-Mex Fuel Co.....	1	1		25
Piedras Development Co.....	1			25
Cia. Petrolera Las Brujas.....		2		
Producers Oil Co. of Tex.....	2			6,500
Cia. Petrolifers, S. A.....		2		
English Oil Co. of N. Y.....	1	3	1	
Cia. Petroleo Mexicana.....	1	1		75
Excelsior-Panuco Oil Co.....	1			1,000
Mexican Gulf Oil Co.....	4	1	1	12,000
Scottish Mexican Pet. Co.....	1			250
M. A. Spellacy.....	1			1,000
East Coast Oil Co.....	7	3		8,000
Mexican Development Co.....				
Tal Vez Oil Co.....	1		1	1,500
Victoria Oil Co.....		3		
Heradura Oil Co.....	1		1	
R. E. Brooks.....		1		
Cia. Petroleo Maritima.....		1		
La Bonanza Pet. Co.....	1	1		200
New Bonanza Oil Co.....		1		
Cia. Abastecedora de Petroleo y Aceite.....		2		
Yisarrí & Blanco.....		1		
Cia. Petroleo Mexicana La Nacional		1		
Cia. Petroleo Los Perforadores.....		1		
Cia. Indio de Petroleo.....		1		
Los Dos Estrellas Oil Co.....		1		
Cia. Petroleo La Oxaquena.....		1		
Cia. Petroleo Panuco-Maugaubes.....		2		
George Harmon.....		1		
Alamo Oil Co.....		1		
Cia. Hispania-Mexicana.....		1		
Panuco-Topila Pet. Co.....		1		
El Vado Petroleum Co.....		2		
Panuco Valley Oil Fields Ltd.....	1			2,000
Pan-American Oil Co.....		1		
Fenix Oil Co.....		1		

Summarizing the above, we find that there were in the Mexican fields during the spring of 1915, 53 wells producing, 111 shut in, and 40 abandoned, while the number of wells drilling, shut down or simply located, was 105.

1. *Markets*

All oil exported from Mexico goes by tank ships. During normal times most of the Mexican railroads are equipped for oil burning, and obtain their fuel from the Ebano field, or from the several refineries at Tampico or Miniatitlan after it has been "topped." By far the greater proportion of all Mexican oil which is exported goes to United States ports, where it is "topped" and the residue is sold as fuel or for paving purposes. During 1914, shipments of Mexican oil were as follows:

Destination	Barrels (42 Gal.)
United States.....	15,476,727
Mexican Coastwise.....	4,510,061
South America.....	195,138
Continental Europe.....	365,205
United Kingdom.....	67,780
Panama.....	46,446
Cuba.....	20,000
Total clearances.....	20,674,357

Refineries.—There are in Mexico the following refineries which supply the domestic market with refined oils:

	Daily Capacity, Barrels
Standard Oil Co. of New Jersey at Tampico...	4,000
Waters-Pierce.....	Small
Pearson (Mex. Eagle) (Eventually 25,000), Tampico.....	12,500
Pearson (Mex. Eagle) at Minatitlan.....	15,000 to 25,000
Huasteca Pet. Co. at Tampico.....	10,000
Huasteca Pet. Co. at Ebano Asphalt Plant....	

2. *Quality*

The oil produced in these fields is divided roughly as to quality into two classes—one heavy gravity, running from 10 to 14° Bé (0.993 to 0.973 sp. gr.); and one relatively lighter, running from 18 to 22° Bé (0.947 to 0.922 sp. gr.). A small quantity of very light crude oil, of 30 to 40° Bé. (0.840 sp. gr.), has been discovered, but no promising developments of it are assured.

The distillation analysis of this oil yields the following percentage of refined-oil products, compared with Illinois and Oklahoma crude oil:

	Yields of Finished Products from Mexican Crude Gravity 26° B ₆ Per Cent.	Yields of Finished Products from Illinois and Oklahoma Crude, Process No. 3, Per Cent.
Benzine.....	7.14	21.00
W. W. Burning Oil.....	21.42	18.38
Gas oil.....	21.42	28.81
Neutral Distillate for Red and Miscel- laneous Lubricating Oil.....	11.71	7.68
Asphalt.....	32.16	17.62
Process Loss.....	6.15	6.51
	100.00	100.00

And the following analyses of characteristic components and heat value are reported by Messrs. Dow & Smith, Chemical Engineers, New York City:

REPORT OF ANALYSIS NO. 6804, JUNE 11, 1912.

Specific gravity at 60°F.....	0.9115
Equivalent to.....	23.8°Bé
Flash, N. Y. State open tester.....	77°F.
Flash, N. Y. State closed tester.....	65°F.
Fire test.....	120°F.
Boiling point (distillation begins).....	175°F.
Water.....	Trace
Viscosity at 32°F. (Engler 1st 50 c.c.), sec.....	1,980
Viscosity at 72°F. (Engler 1st c.c.), sec.....	298
Loss on heating 20 g. 5 hr. at 325°F., per cent.....	26.8
Fixed carbon, per cent.....	7.23
Paraffine scale, per cent.....	2.25
"Asphalt Contents" 50 pen. (slow evaporation 32 hr. in air bath at 300 to 400°F.), per cent.....	59.5
"Asphalt Contents" 50 pen. (quick evaporation by burning), per cent.....	52.0
B.t.u.....	18,493

Ultimate Analysis

	Per Cent.
Carbon.....	82.83
Hydrogen.....	12.19
Oxygen (by difference).....	0.43
Nitrogen.....	1.72
Sulphur.....	2.83

REPORT OF ANALYSIS, NO. 7288—CRUDE MEXICAN OIL

Specific gravity at 60°F.....	0.985
Equivalent to.....	12.1°Bé
Flash (closed tester).....	157°F.
Fire test.....	225°F.
Sulphur, per cent.....	4.20
Scale paraffine, per cent.....	0.518

Calorific value.....	18,227 B.t.u.
Viscosity at 60°F.....	Too thick to take
Viscosity at 100°F.....	Too thick to take
Viscosity at 212°F. (200 c.c. Engler Viscometer)....	615 sec.
Flows readily at.....	150°F.

Fractional Distillation

	Per Cent.	Products,
1st drop, 140°F.		
140 to 302°F.....	4	naphtha distillate
302 to 545°F.....	20	burning oil distillate
545 to 600°F.....	10	cracking commenced
600 to 620°F.....	15	cracking continued
Residue and loss.....	50	cracking badly, temperature fell to 605°F.
Penetration at 77°F. of residue from fractional distillation.....	60	

Asphalt Contents by Evaporation

100 penetration at 77°F., per cent.....	70.1
Ductility of ditto, cm.....	100.0
50 penetration at 77°F., per cent.....	65.7
Ductility of ditto, cm.....	100.0

The following are analyses of the Huasteca oil:

HUASTECA OIL,

CASIANO FIELD, WELL No. 7,

SAMPLE I

Gravity 19.4° Bé.....	Pour, 0
Flash, 80°F.....	B. S., 3 per cent.
Viscosity, 450 B. O. T.....	B.t.u. per pound, 19,124

Fractional Distillation

Temperature	Per Cent.	Gravity, Degrees Baumé
Over from 259 to 385°F.....	10	58.7
Over from 385 to 493°F.....	10	
Over from 493 to 500°F.....	1	45.9
Total		
Distillates.....	21	51.5
Residue.....	78	13.5; pour, 40
Loss.....	1	
	100	

SAMPLE II

Gravity 19.3° Bé.....	Pour, 0
Flash, 96.....	B. S., 2 per cent.
Viscosity, 500 B. O. T.....	B.t.u., per pound, 19,160

Fractional Distillation

Temperature	Per Cent.	Gravity, Degrees Baum
Over from 244 to 400°F.....	10.0	57.8
Over from 400 to 493°F.....	10.0	
Over from 493 to 500°F.....	0.6	45.1
Total:		
Distillates.....	20.6	50.8
Residue.....	78.5	13.4
Loss.....	0.9	
	100.00	

3. Costs and Prices

The cost of production of oil (of about 12° Bé.) at Panuco at present is about as follows:

Cost per Barrel of 42 Gal.	U. S. Gold, Cents
Bar dues (*50c. Mex. per Metric ton or 6.6 bbl. $\pm$ av.)	3.8
Production tax (30c. U. S. per Metric ton).....	4.5
One-tenth Royalty (based on an arbitrary value of 50c. U. S. per barrel).....	5.0
Transportation by barge, Panuco to Tampico.....	6.0
Lighterage (terminal to shipside at Tampico).....	2.0
Production expenses (estimated).....	10.0
Cost of barrel of oil, f.o.b. ship at Tampico.....	31.3

(*NOTE: This is Mexican gold with an exchange value of 2:1).

The cost of transportation by tank ship to United States ports may be estimated as 10c. per ton per day. As the voyage consumes from 3 to 10 days, depending upon conditions and the port to which the oil is consigned, the total cost to destination will vary from 35.5 to 45.3c. per barrel. Costs from Tuxpam will differ with differing producing conditions in the field; pipe line instead of barge transportation, and no bar dues; but with a slightly longer sea voyage for tankers.

The official market price during 1914 for Panuco heavy oil f.o.b. ship at Tampico, ranged from 40c. a barrel in January to 20c. (U. S. gold) at the close of the year. Evidently some oil was sold at less than the cost of production, in order to keep equipment in use, and to take care of production which could not be shut in. Very little oil is sold at the wells in the Mexican fields, although a few sales have been reported at Topila and Panuco.

The lighter oils (19 to 27° Bé.) from the southern districts, are all sold under contract at higher prices than the Panuco oil. Casiano oil is valued at from 50 to 60c. f.o.b. ships at Tampico, which would make its value at United States ports from 55 to 75c. per barrel. The oil from the properties of the same company at Jan Felipe and Cerro Azul is reported to be as light as 27° Bé. The oil from the Alamo lease of the Penn-Mex Fuel Company is about 24° Bé., which is slightly higher than that from the Potrero del Lano lease of the Mexican Eagle Petroleum Co.

4. *Drilling*

Drilling is carried on in these fields with heavy combination rigs, usually of the California type. The rotary is used through the soft upper marls (Mendez or Los Esteros beds) into the upper portion of the San Felipe formation. Here the casing is set on top of some hard limestone shell, and drilling is continued with percussion tools through the San Felipe and well into the massive Tamasopa lime, unless oil is found sooner. In most of the large wells the oil flow has been struck in broken blue shale or limestone, a short distance above the top of the Tamasopa. In others, the oil is found in the upper few feet of the Tamasopa. There are no defined oil "pays" which can be correlated with each other from one field to another, or even from one well to another in the same field. The oil occurs in fractured zones, in which solution channels caused by circulating underground waters are frequent. This condition has led to the generalization, based on the experience of operators in these fields, that all the big wells get their oil "high up;" or in other words at a shallower depth than the average for the locality.

Drilling is very expensive in these fields. An operator expecting to drill must estimate that his first wells, if wild-cats, will cost in the neighborhood of \$65,000 each, exclusive of the leasing expenses. This excess outlay may be divided under the heads of high wages; the establishment of camp and commissary; the transportation and duties on drilling materials; the delays due to accidents and waiting for materials, or the cost of keeping a large stock of extra parts to prevent such delays; road building and rights-of-way for roads and water lines; the high cost of fuel, for wild-cat wells; and the cost of keeping up communication with Tampico, by means of launches and horses.

The difficulties and expenses of drilling are affected by geological conditions also. Along the coast line, and the shores of Lake Tamiahua, there are deposits of varying thickness, and of Quarternary and Recent age. This is also true in the Panuco field, where they are about 100 ft. thick, and obscure all evidence of structure and intrusion. The Mendez beds vary considerably in thickness within the same district; and the same appears to be true of the San Felipe series. There are no horizons within either of these formations which can be correlated, even between two nearby wells. And owing to the difficulty in determining exactly from a well record the top of the Tamasopa limestone, due to the gradual transition from the San Felipe series to a massive limestone phase near the bottom, it is impossible to ascertain positively the extent and position of such unconformities as probably exist. This is even more difficult because of the lack of cooperation upon the part of the various companies doing geological work in these fields. The isolated location of the various districts permits a degree of secrecy possible in few other

fields; and several of the large companies have carried this policy to an unjustifiable extreme. This does not apply, however, to the Panuco and Topila districts.

The entire oil-producing area is crossed in various directions by igneous dikes and intrusions. These are generally in the form of basalts

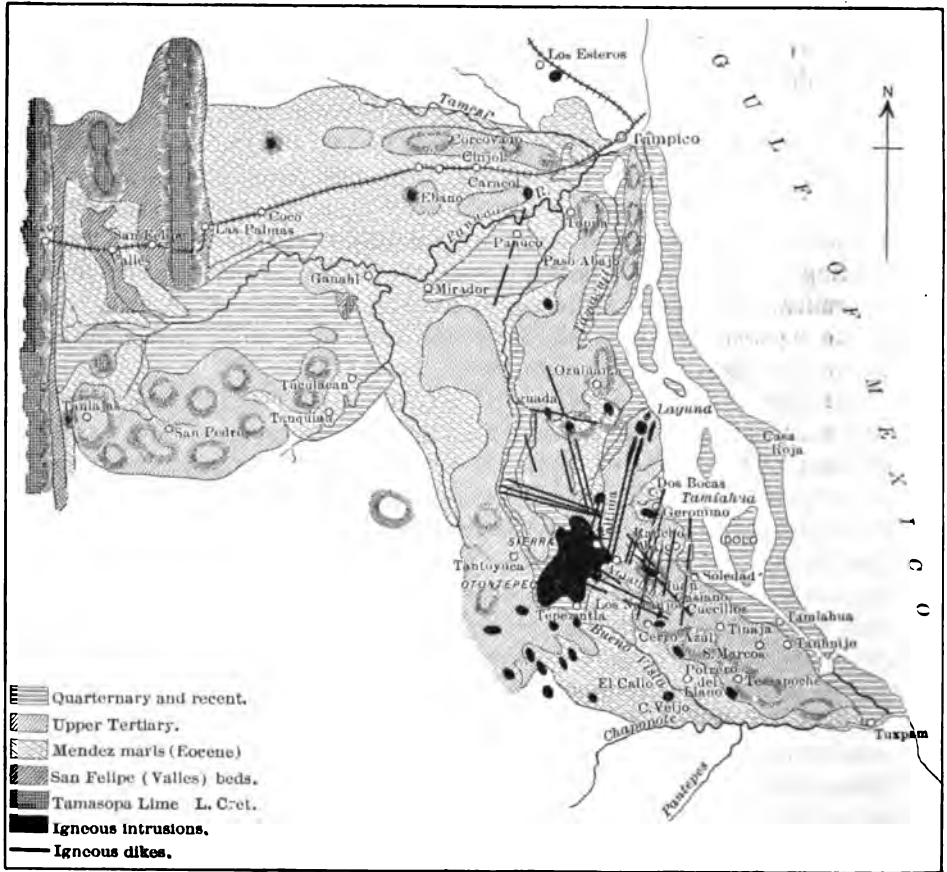


FIG. 2.—GENERALIZED SKETCH OF THE MEXICAN OIL FIELDS, SHOWING AREAL GEOLOGY, LOCATION OF MAIN BASALTIC INTRUSIONS, AND STRIKE OF THE MAIN DIKES IN THE CENTRAL DISTRICT.

and diorites, and at various places occur as very large intrusions. The largest of these is the Sierra Otontepec, shown in Fig. 2.

As there are no common carrier pipe lines, and all companies owning their own lines have enough oil of their own to fill contracts, there is no sale for oil in the field. Therefore the small operator sometimes gets a white elephant on his hands by drilling a good well in a wild-cat district. He may sell outright at considerably less than the well is worth, if he can

find a purchaser; or promote a company of his own to handle the well; or enter into an alliance with others who will furnish money to build pipe lines to seaboard, erect a terminal, charter or build tank ships, and finally market the oil in the United States or elsewhere. As matters stand at present, with so many wells shut in, the finding of the oil is the least important step, particularly if there is not enough territory leased in the vicinity to protect such a well, and justify capitalists in undertaking the further development of the project.

IV. GEOLOGY

1. *Stratigraphy*

In the Mexican fields four distinct formations are encountered: (1) an upper series of fossiliferous Tertiary sandy limes and sandstones, interbedded with limy and sandy clays, the beds varying in thickness from 600 to 1,300 ft.; (2) an intermediate section, 2,000 to 3,500 ft. thick, of gray marls and shales (called Mendez marls, or Los Esteros beds), the upper portion of which has been indentified as Eocene Tertiary which give place in their lower portion to (3) the San Felipe or Valles beds of limestone shells alternating with blue and brown shales 200 to 700 ft. thick. These lie upon (4) a massive blue-gray (Tamasopa) limestone formation, at least 3,000 ft. thick, fossiliferous in its upper portion, of Lower Cretaceous age. In some areas, the upper Tertiary limestones have been eroded away, and the surface formation consists of marls (see Fig. 2).

a. The Upper Tertiary.—The upper Tertiary formations have been divided by some geologists into the western or Tanlajas series, and the eastern or Ozuluama and Temapache series. While they keep their general character as marine deposits of rapidly alternating sandy limestones and clays, and local beds of fossils such as the beds of nummulites and orbitoides in the central districts can be correlated for short distances, yet such correlations cannot be carried from one area into another without much care.

In the western or Tanlajas district these beds average about 1,100 ft. in thickness, while in the easternmost wells—those at Tanguijo—they are at least 1,200 ft. thick. These upper Tertiaries, however, practically cover the southern and central portions of the Mexican field, except where river valleys have exposed the underlying marls by erosion, or where large basaltic intrusions have taken their place.

These upper Tertiaries have been found oil-bearing at one locality only—Tanguijo. Here the Aguila company has drilled a number of wells, which are capable of pumping about 500 bbl. per day. The oil is 21°Bé. and the wells decline rapidly. Near the bottom of the upper Tertiary section in the Tuxpam district, a dark bituminous sand bed is reported. This does not appear in the northern section. However, owing to the

friability of these beds, and the prevalence of strong jointing, one finds a great many asphaltic seepages exuding from them at the surface. In most cases this is due to the fracturing of the underlying beds, and the intrusion of dikes; the oil finding an easy egress through these upper beds. It is believed by some that the oil found in these beds at Tanguijo has found its way there through migration from lower formations, through fractures. I regard this as very probable. The size of these wells and their behavior hardly encourages prospecting in these beds for the development of commercial production, especially while wells in other districts are so prolific.

While there are numerous exposures of these upper Tertiaries in the central and southern districts, and good clinometer readings can be obtained at many points, yet such data frequently prove misleading. The absence of good key horizons makes it difficult to locate fault lines, or to determine their throw. It is also frequently impossible to work out the details of the monocline structure found in these beds; but since these minor irregularities are probably of little importance, this difficulty is not of great consequence. In other words, intrusions and fractures, channelling in the oil-bearing limestone, and the effect of sedimentary gradients and unconformities are all of so much greater importance in these fields in their influence on oil accumulation, that only the large well-marked structural features need be considered, where these other factors are not known to exist.

b. The Mendez Marls.—The Mendez marls consist of a very uniform deposit of gray to blue shales and marls. In regions of steep folding, these often show bold jointing near the surface. There is practically no change in their lithological character from top to bottom. They average from 2,000 to 3,500 ft. in thickness. In the Huasteca country, they are nearer the former thickness. A few irregular beds of sandy limestone are reported in this formation, but they are not persistent. However, there are frequently small quantities of heavy oil immediately below them, although never enough to sustain production of commercial importance.

The apparent thickening or thinning of these beds within short distances is sometimes accounted for by faulting of the formations. Or a local steepening of the dip may cause an apparent thickening. The upper limestone lenses in the San Felipe beds likewise come in and go out, and seem to indicate unconformity; so that the thickening of the Mendez marl beds is counterbalanced by a smaller thickness of the lower San Felipe series, the depth to the top of the Tamasopa limestone remaining practically constant. Even this interval, however, is not always constant in the same district, for the reason that there is more or less unconformity between the top of the Tamasopa and the overlying San Felipe shells.

c. *The San Felipe Formation*.—This may be described as a transition series between the upper Mendez marls and shales and the underlying massive Tamasopa limestone. It begins with an occasional thin limestone shell. These increase with depth in number and thickness, being interbedded with blue shales which conversely decrease in thickness downward until the series gives place to massive limestone. These beds apparently vary in thickness from about 300 to as much as 800 ft. Since drillers frequently do not begin to record these shells until they are about ready to set the casing, it is impossible to estimate the thickness of the series in every well merely by the well log.

These San Felipe beds are frequently the oil-bearing formation in the Mexican fields, particularly for the largest wells. This is because of the well-known occurrence of strong fracturing, with which large oil accumulations are associated. Oil is found in such a fracture as high as it can go; and any well drilling into such a "fissure" will encounter its big flow "high up," in the terms of the operator; the marls finally proving an effective seal against its escape above a reservoir in the San Felipe. Among others, the following "gushers" never reached the Tamasopa lime: Juan Casiano No. 6 and No. 7; Pearson's Los Naranjos No. 1 and No. 4; La Corona No. 5 at Panuco, and Spellacy No. 1 at Panuco. In such cases, the oil is usually found in broken shale or channelled limestone in the San Felipe formation. However, since this channelling is probably more frequent at the base of the series, where there is considerable unconformity, the top of the Tamasopa lime has come to be looked upon as the "pay" formation.

While such faulting or fracturing extends to the surface through the Mendez beds, as indicated by dikes and seepages, yet it is perfectly evident that as these upper formations contain no porous beds, no oil accumulations can be looked for above the harder San Felipe. As these soft marls have undoubtedly been pinched together after disturbance took place, they have in many instances effectually sealed any outlet except such passageways as those kept open by igneous intrusions. However, such seepages may travel laterally for considerable distances along joint cracks in the harder upper Tertiaries, before finding an outlet to the surface—making the successful location of wells problematic if based upon no other evidence than a seepage of *chapopote*.

d. *The Tamasopa Limestone*.—Jeffreys says that the limestones of the San Felipe series can be distinguished from the lower Tamasopa lime in that (1) they contain no fossils; (2) they are not massive; and (3) their texture is generally of more even grain than that of the Tamasopa. These differences, however, are not always easy to detect in a well sample. The Tamasopa is a compact, hard, gray, crystalline limestone, fossiliferous near the top. It has never been drilled through, and no productive horizon is known far below the top. There are no oil seepages or evi-

dences of hydrocarbon contents at the outcrop, far from the contact of overlying formations. There appears as yet no conclusive evidence that the oil in the Mexican fields had its origin in the beds of the massive Tamasopa limestone, as claimed by some geologists. The very nature of such beds, especially considering the subsequent period of erosion, is against such an assumption. On the contrary, there is more evidence here than in many other fields pointing to the probability of its having migrated from the overlying marine marls. This will be discussed in another paragraph.

2. *Structure*

All the large wells drilled up to the present are located where there exists a significant combination of both favorable anticlinal or dome structure with pronounced fracturing of the formations. These fractures (frequently faults of relatively small throw) are usually accompanied by basaltic intrusions, and seepages of asphalt and gas. In the Panuco field, all surface evidence both of structure and intrusion, has been obscured (except at one point north of the river in the Tampalache area) by about 100 ft. of alluvial sediments; but drilling has shown that the same conditions exist and have influenced the oil accumulation here as in other parts of the Mexican fields (see Figs. 3 and 4).

There has also been faulting in connection with some of these anticlinal structures, especially those nearer the mountains (Fig. 5). These folds become broader and less frequent to the eastward (see Fig. 5). However, other forces caused a number of relatively well-marked folds in the vicinity of Otontepec, such as those at Potrero del Llano and Los Naranjos. This folding was caused by lateral thrust and probably certain vertical stresses incidental to the formation of the Sierra Madre mountains to the west. These in turn set up other lines of weakness in the formations, through which during late Tertiary time other igneous rocks were intruded. This is shown by Fig. 6, which is a map of the central part of the fields in which many of the main basalt dikes have been located by the author.

A study of this map will reveal a number of interesting relations, for instance, the general agreement between the strike of the sedimentary formations and that of the main dikes in the coastal portion of the fields. A reference to the sketch map (Fig. 2) showing the general areal geology of the Mexican field, fails to reveal any locality where the Tamasopa lime or even the San Felipe beds have been thrust up to the surface by intrusives, as claimed by some of the earlier writers. The writer knows of no instance where any pronounced doming has been caused by the upthrust of dikes or so-called "plugs" of basalt. Some very local distortion and faulting has been caused at certain places; but such instances are balanced by others in which the sedimentaries actually dip toward

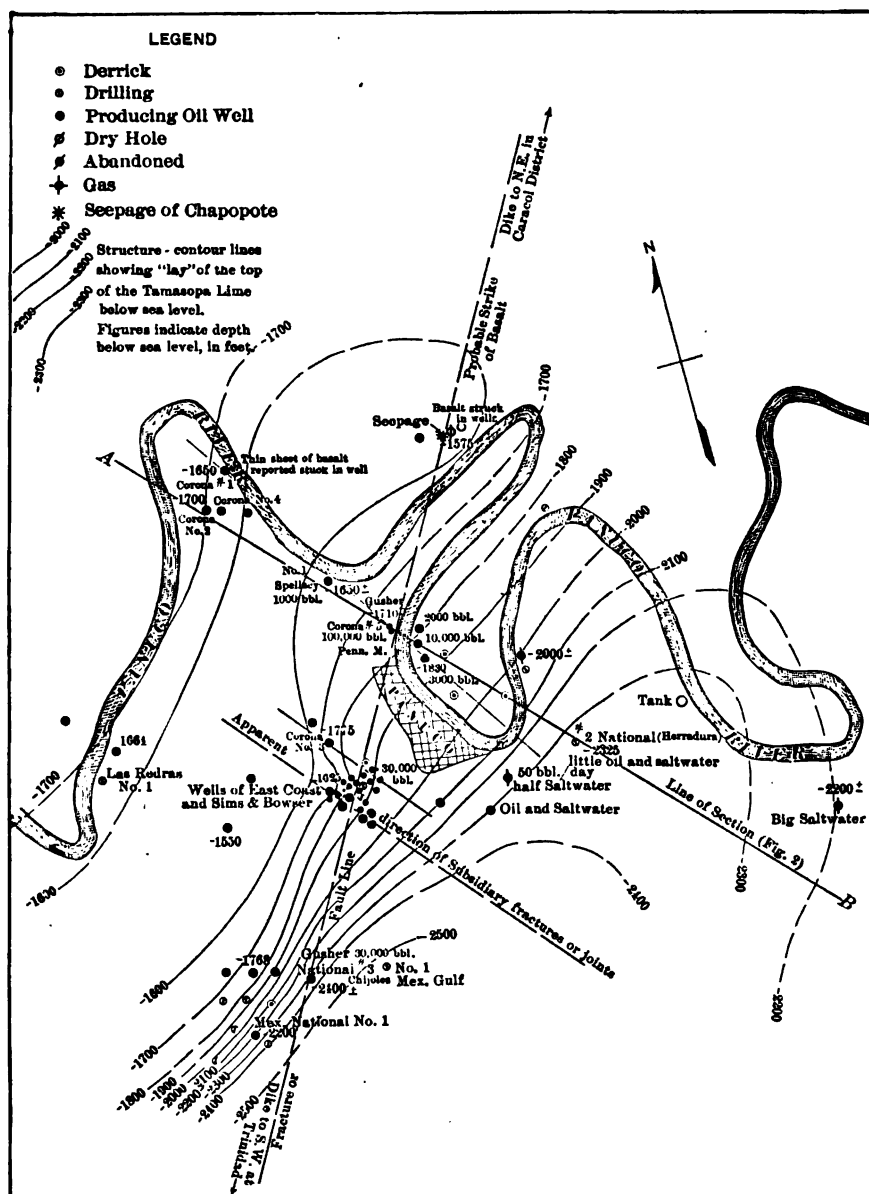


FIG. 3.—MAP OF THE PANUCO OIL POOL, SHOWING "LAY" OF THE PROBABLE TOP OF THE TAMASOPA LIME, AS INDICATED BY WELL LOGS; AND ALSO THE LOCATION OF PRINCIPAL WELLS.

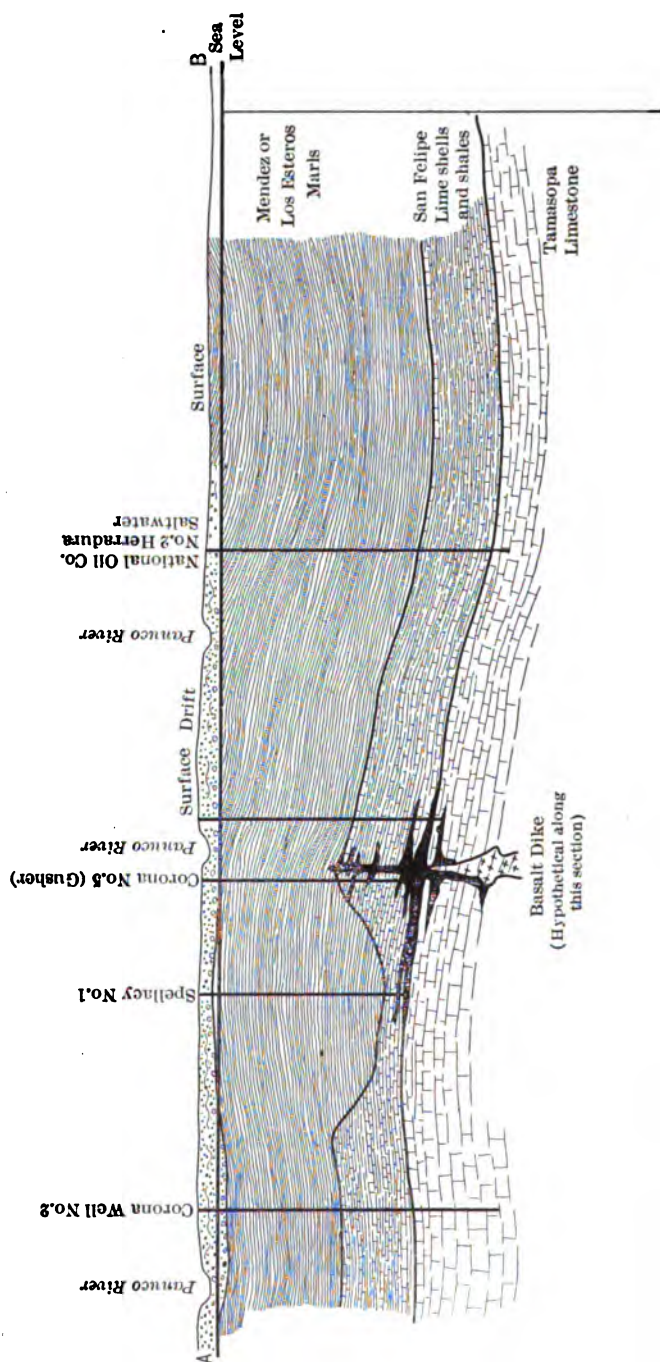


FIG. 4.—HYPOTHETICAL SECTION THROUGH THE PANUCO FIELD, MEXICO, ALONG THE LINE A-B (FIG. 3), SHOWING ANTICLINAL TERRACE AND FRACTURE.

(Note: The character of the unconformity as shown between the marls and the San Felipe beds is tentative.)

large igneous bodies from all sides. That is to say, shrinkage due to cooling has more than offset any "doming" effect.

In common with other fields where the surface beds are friable, the local dip of such beds as measured by a clinometer is apt to be much greater (especially near intrusives) than that of the underlying formations. Additional drilling has failed to corroborate the "nail" theory of the effect of igneous plugs or other intrusions upon the structure of surrounding sedimentaries, in the Mexican field.

A reference to Fig. 7 will show that some secondary folding by lateral thrust has been caused by the intrusion of the Sierra Otontepec—the largest igneous mass in the field west of the mountains.

Igneous Intrusions.—Again referring to Fig. 6, it will be seen that the fields at Juan Casiano, Los Naranjos, Dos Bocas and Panuco are all located at the intersection of strong fractures, where such intersections occur on anticlinal folds. Access to the well records of certain operating companies would undoubtedly reveal interesting data with regard to the relation of the accumulation of oil under these conditions to possible fault displacement. Where the marls are exposed at the surface, as at

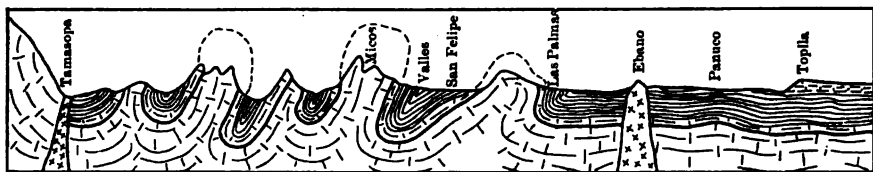


FIG. 5.—GENERALIZED SECTION EAST AND WEST THROUGH NORTHERN PART OF THE OIL FIELDS. AFTER JEFFREYS.

Los Naranjos and Dos Bocas, it is impossible to determine local faults from surface indications, unless they have considerable throw or linear extent, enough to make it possible to locate them by inference. Nor is it at all easy to determine the throw of such possible faults, even where the harder Tertiary beds are found at the surface, due to the lack of well-defined key horizons. However, sufficiently detailed work will usually give enough clues to their existence, especially after a knowledge of conditions in the surrounding areas has been gained.

Intersections of strong fractures are frequently accompanied and marked at the surface by conical basalt peaks, which usually represent the "mushrooming" of an igneous neck intrusion. Wells drilled close to the contact at several of these conical hills have disproved the theory, advanced by one geologist, that they were "plugs" whose conical shape persisted at depth. Such wells have frequently started close to the contact and have been drilled into the oil formation at more than 2,000 ft. in depth, without encountering any further basalt, or any violent distortion. These intrusions and accompanying flows apparently follow

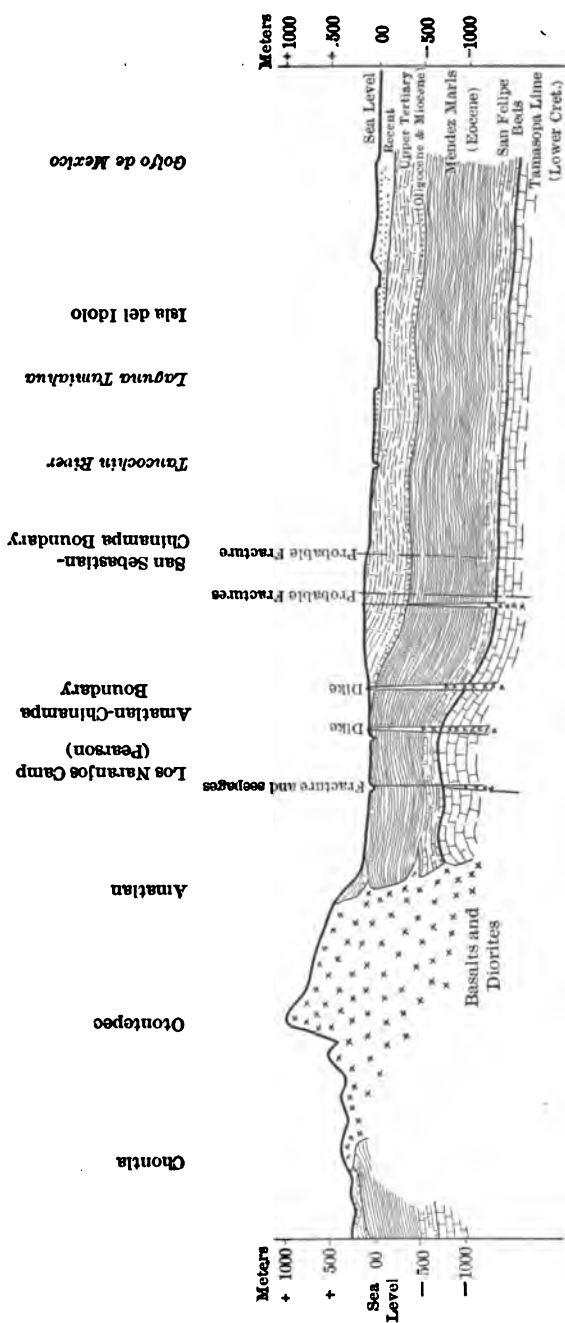


FIG. 7.—DIAGRAMMATIC GEOLOGICAL SECTION ACROSS LOWER PART OF FIG. 6.

such intersections, not filled with basalt to the surface, offered a channel for the circulation of underground water more or less hot or highly mineralized, which metamorphosed the sedimentary formations in the immediate vicinity. These formations would thus resist erosion much more readily than the softer limes and sands about the channel. One notable example of this is Cerro de Zaragosa, between Amatlan and Zacamixtle. This has every appearance of being a typical basalt peak, yet examination failed to show any basalt on its sides, which are composed of upper Tertiary formations. And yet the peak is directly in line with a main series of dikes extending from near Dos Bocas to Zacamixtle, through Juan Casiano and Los Naranjos. (Fig. 6.)



FIG. 8.—CERRO TANINUL, A CONICAL BASALT PEAK TYPICAL OF THOSE OCCURRING AT MANY POINTS IN THE MEXICAN OIL FIELD.

Another example is the eastern peak of the two on the edge of Aguada, called Taninul. The western one is composed of basalt (Fig. 8), while the smaller eastern peak shows only large blocks of a metamorphosed marl containing numerous small siliceous nodules. This peculiar rock is found at a number of places in the vicinity of large intrusions of basalt, and has been caused by the circulation of underground currents, probably both hot and more or less mineralized, as pyrite is frequently found nearby. There are also numerous smaller peaks and conical knolls throughout the field.

3. *Source of the Oil*

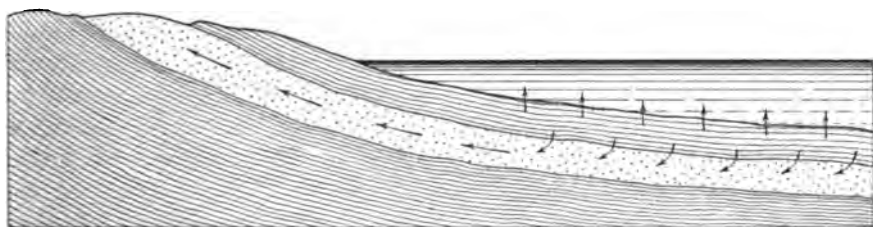
In a consideration of the source of the oil found in the wells in the Mexican fields, the following facts hold true at all points:

1. The oil is found in a porous and usually fractured limestone (sometimes shale) near the top of the Tamasopa limestone formation. Although a few shows, and, in some cases, large quantities of salt water, have been encountered, no oil in large quantities has ever been found as yet by drilling deeper into the lime.

2. Oil is also found, particularly in the "gusher" wells, in the broken lime shells and blue shales of the San Felipe series, usually under conditions indicating strong fracturing and jointing.

3. Oil is not found in quantity in the homogeneous marls overlying the San Felipe, although these marls are more or less petroliferous throughout, as shown by drilling. However, in drilling near dikes and fractures when seepages occur at the surface, shows of gas and heavy oil are often encountered in the hole all the way down.

4. These marls were laid down upon already compacted limestone beds of great thickness (Tamasopa) in which, so far as now proved, no



After King, 19th Annual Report, U. S. Geological Survey, part 2, p. 80.

FIG. 9.—DIAGRAMMATIC SECTION SHOWING THE FLOW OF CONNATE WATER, OIL, AND GAS, DUE TO CONSOLIDATION OF SEDIMENTS.

porous horizon existed. It is not considered probable that oil could have been formed within massive beds of marine limestone such as these.

5. Any oil coming from below the limestone must have been liberated by fracturing and intrusion and subsequently caught and held by the denser marls, pinching together and retaining such oil in the more easily shattered and hence more porous San Felipe and upper Tamasopa lime. Circulation of underground waters along the unconformity on top of the Tamasopa may have aided in forming reservoirs for the oil. There is considerable evidence of these channels in some wells, but they may have been formed after fracturing had promoted circulation.

This (5) presupposes either the inorganic origin of oil, or the existence of enormous previous accumulations of it in formations at great depth—below the Tamasopa. The latter alternative does not seem reasonable, because such previous accumulations would already have been more or less localized in certain areas, and when allowed to filter upwards through say 10,000 ft. of limestone after later fracturing, would have shown much more varied composition, and would have been restricted to more local

and erratic areas in the upper formations in the Mexican fields, instead of being so generally distributed under so nearly the same conditions.

The author holds it much more reasonable to assume that the oil had its origin in the petroliferous marine marls of the Mendez formation. The compacting of these marls under a constantly increasing overburden, as demonstrated by King (Fig. 9), would have forced the liquid content into any more porous bed existing below. It must be remembered that such marls are not impervious to the movement of fluids at the time of compacting, although they gradually become relatively so. As there is evidence of much channelling by hot water along the top of the Tamasopa, these channels offered both a course for the necessary movement, and the porosity for concentrating such oil as found its way into them. As stated by Roswell H. Johnson, "If beds of less compressible material (porous) meet or underlie the shales in question, there will frequently be a lateral movement along such beds to some point where upward movement will be resumed." There may be even today such compacting and lateral movement still going on along the shores of the Gulf of Mexico, which would be most actively manifested along fracture and fault lines where circulation upward could be resumed most freely. The greater the opportunity for this movement, while at the same time there was a chance for the segregation and retention of the oil, the larger and more extensive would be the final accumulations. Later movement along bedding planes of the marls probably found ideal catchment areas in fractures which had been formed in the meantime, and added to the accumulation of oil already there. This may account for the different composition of oils in the different districts.

Many phenomena noticed in connection with the production of oil in these fields are worthy of study. Employees of the Mexican Eagle Petroleum Co. report that when sudden barometric changes occur, such as those accompanying the frequent "northers" on the Gulf coast, the closed pressure of the Potrero del Llano well No. 4 increased as much as 100 lb., while nearby wells also show an increased pressure of smaller extent. If true, whether this indicates a connection between the underground reservoir and the Gulf through channels or fractures in the hard lower beds, or (what is more likely) direct connection of the oil and water table to the surface through fractures kept open by intrusives, is a question upon which more data are needed.

a. Basalt.—A reference to Fig. 2 will show that while the master intrusion in the central part of the Mexican fields is the Sierra Otontepec, and that radiating and tangential dikes in the immediate vicinity may be due to secondary fracturing caused by this large intrusion and its subsequent shrinkage; yet in general this represents only an exceptionally large example of a type common throughout the field.

So far as worked out by the author (Fig. 6) the long fracture lines

(usually filled by basalt dikes) were the primary phenomena. These are intersected at various angles by shorter fractures which resemble strong joint cracks at the surface, and which are less frequently filled by igneous intrusions. These are sometimes filled by sandstone dikes, as in the vicinity of Juan Casiano, and are in places indicated by short series of asphalt seepages following a general direction. As at such intersections there are so frequently found conical basalt peaks, or even hills of metamorphosed sediments, one soon comes to the conclusion that these knobs were merely phenomena incidental to and accompanying the main dikes. There is a group of such hills at La Pithaya and Juan Casiano.



FIG. 10.—OLIVINE-DOLERITE. MAGNIFICATION ABOUT 22 DIAMETERS.

Augite, olivine, and magnetite in well-formed crystals, and evidently crystallized before the feldspar. Labradorite in hemimorphic lath-shaped crystals surrounding the other constituents.

In neither case has the size of any one of these peaks, nor the distortion of the sedimentaries accompanying it, been enough to cause fractures of such length as are known to extend from these localities both north and south. The immediate presumption after studying these localities is that such fractures are the main fact to be taken into consideration.

Now, it will be noticed in Fig. 6 that Otontepec lies at the intersection of the two main series of dikes found in the central district—one extending approximately 20° W. of N. toward Panuco, and the other at about the same angle east of north toward Dos Bocas and the Gulf. These fractures are so persistent that the same conclusion appears to follow as was stated in the preceding paragraph, viz., that Otontepec is merely



FIG. 11.—SPECIMEN FROM A SMALL DIKE. MAGNIFICATION 40 DIAMETERS.
Larger proportion of augite than Figs. 10 or 12.



FIG. 12.—OLIVINE-DOLERITE. MAGNIFICATION 22 DIAMETERS. SAME AS FIG. 10.



FIG. 13.—BASALT DIKE APPEARING AT THE SURFACE IN THE MEXICAN FIELDS, NEAR OZULUAMA.



FIG. 14.—PORTION OF A LARGE SEEPAGE OF "CHAPOPOTE" IN THE CENTRAL DISTRICT OF THE MEXICAN OIL FIELDS.

Notice the remains of the cow which became entrapped in the bed of asphalt.

an intrusion which represented the point of greatest weakness in the sedimentary rocks. It probably was formed by the coalescing of a number of necks which found their way to the surface as did the smaller ones found at other points. A study of the districts south and west of Oton-tepec would probably yield confirmatory evidence. As it is, there are no published data on these districts.

The accompanying sections (Figs. 10, 11, 12) were taken from specimens of basalt dikes in this central district.

b. Seepages.—Seepages of asphalt, sometimes accompanied by gas and occasionally by sulphurous brine, are found throughout the Mexican field, and, with the relatively horizontal strata and igneous intrusions, constitute a unique feature of this oil field. Several of the largest groups of seepages (Fig. 6) are found on anticlines, and most of them are near dikes or intrusions which reach the surface. Others, however, appear to exude from the surface formations where there is apparently no structural reason for oil accumulation below. However, experience and inference have led to the general belief on the part of geologists that in most instances seepages indicate an intrusion of igneous rock in the vicinity not far from the surface (Fig. 14).

It is common to find short lines of seepages intersecting a main dike at a high angle (Fig. 6). A series of such short lines of seepages, each at an angle to the strike of that district, yet taken together following the strike of a dike, is often evidence by which a fracture can be projected from another locality. The shorter intersecting fractures probably in most cases represent joint cracks due to local pressure. The photograph (Fig. 13) shows a spur of basalt from the dike following such a direction.

Seepages are also found exuding from basalt high on the Sierra Oton-tepec, at a considerable distance from the contact of the sedimentaries. As there are many open channels through the basalt which would obviously offer less resistance to the passage of oil than would the overlying Mendez marls, these apparently anomalous occurrences are not so in reality. Basalt dikes cannot be looked upon as barriers to the movement of oil, except so far as they accompany fractures which tend to entrap it. Such migration may be from several directions.

A study of the central district shows a significant resemblance between the fractures located and a series of joint or shrinkage cracks which might be due to a compacting and drying of the sedimentaries. A remarkable persistence of roughly triangular areas inclosed by these fractures, and the fact that their persistence in certain general directions has some relation to the direction of the main stresses at the time of the Sierra Madre uplift, suggest that such general jointing or shrinkage cracks may have been the primary cause of the lines of weakness which were followed later by intrusions.

4. *Limits to Prospective Fields*

In the area between Las Palmas and Micos, some sharp anticlinal structures occur, in one of which several wells have been drilled (San Pedro), with a small production of light oil (45° Bé.). This, no doubt, will be further prospected; but as the production per well will probably continue small, and as operating is very expensive, it will probably not pay to go after such risky prospects when the low-gravity oils are so plentiful. The cost of producing this higher-grade oil will make it more profitable to import such refined products as cannot be made from the lower-gravity oils of the eastern fields.

The Topila district will probably be developed into an area where the salt-water problem will not be so serious; and probably such areas as Caracol and Los Esteros will be finally developed into better producers, although the first wells have been disappointing. There is then a great area north of Tampico, along the coastal plain between the Panuco River and Texas, in the part of which south of Soto la Marina (San Jose de las Rusias), at any rate, prevailing conditions are similar to those in the southern region. The Dutch Shell interests have large holdings in this northern area. There are many seepages and igneous intrusions in a part of this region.

The district between Las Palmas and Ebano and thence south to Tantoyuca, has not been prospected, owing largely to transportation difficulties. Before the present political difficulties, a railway had been begun, to connect by a short route between Tampico and the city of Mexico. When completed, this will open up the area mentioned. While in general this area does not seem to be so favorable as that east of Otontepec, yet not so much is known of its geology and structure, and it is probable that good pools will be developed, possibly of lighter oil than that of the present fields.

In the extreme eastern belt, along the coast between Tampico and Tuxpam, owing to the fact that the full thickness of the upper Tertiary formations must be passed through, together with considerable thickness of still more recent deposits, drilling at many points will have to be very deep before encountering either the San Felipe beds or the Tamasopa lime. At Tanguijo the upper Tertiary beds are at least 1,200 ft. thick, and a well 4,000 ft. deep had not passed the San Felipe. Moreover, the frequent deficiency of surface indications through the relative scarcity of seepages; the lack of intrusives appearing at the surface; and the possibility of seepages having travelled a considerable distance through the extra thickness of upper formations before appearing at the surface;—are all factors making it difficult to locate wells with the same chance of success as elsewhere. See pages here are not so significant of accumulations or strongly fractured zones directly below, as in regions where the upper

beds are thinner. It is also argued that so far down the dip to the eastward there is more danger of encountering salt water. This is not so dangerous as would at first appear; for oil would be caught in the upper part of fractures, while the water would encroach along the lower portion, and thus "spill" around the entrapped oil, much as happens in the case of encroaching water passing and retaining the oil in the crest of a sharp anticline. Such wells might possibly be shorter lived, but this is theoretical and can only be established by the drill.

V. PROBABLE FUTURE OF THE MEXICAN FIELDS

1. *Evidence of Producing Wells*

The first important producing well in the Mexican fields was the Mexican Petroleum Co.'s Pez No. 1 in the Ebano district. This well has been producing since 1904, and still continues to flow with little diminution at more than 1,000 bbl. per day.

The famous Juan Casiano No. 7 has been producing continuously since September, 1910, and is now making 700,000 bbl. per month, and has approximately 40,000,000 bbl. to its credit.

Nor are these isolated cases. The Pearson No. 4 well at Potrero del Llano has done at least as well as Casiano No. 7 for about the same length of time, and the older wells at Panuco are still good producers in the most closely drilled field of all. While the Chila-Salinas and Topila fields have suffered from the invasion of salt water in the past, yet recent development work indicates that adjoining areas may be better in this respect. The Dos Bocas gusher blew a great crater in the ground, and after producing 200,000 bbl. of oil per day for 57 days, went to salt water. Large quantities of asphalt are still thrown out with the salt water and gas from this hole, and there is reason to believe that if this well had not got beyond control it would have held back the salt water until its oil production would have been much greater in the aggregate than the actual amount expelled during its short life. As already observed, the larger wells in a number of districts are inclined to increase their production after being allowed to flow wide open for a short time. All the evidence of past wells points to the long life and high productivity of most of the districts already tested, and the probability of many more being eventually opened up.

2. *General Summary*

It is believed that Mexico is potentially the second greatest oil-producing country in the world today. The reasons for this belief are summarized below, as well as the causes which will tend to hold back for some time the realization of the possible maximum production:

1. A large number of widely separated areas have been tested and have developed uniformly large wells.

2. With the exception of a few initial wells, none of these areas has been fully developed by the drill. There is thus a considerable acreage undeveloped in already proven territory.

3. Numerous promising localities remain untested in the present fields.

4. Large districts yet untested, such as those north of Tampico and west and northwest of Otontepec offer surface evidence of the same characteristics as the proven fields.

5. The production per well has been large, and the decline slow in most fields.

6. The underground accumulations in fractured zones are capable of yielding a much larger percentage of their petroleum content than is a sand body.

7. More efficient methods of production result from the usual control of each producing district by one large company.

If such districts were located in the United States, they would soon be drilled up to a production such that the price would break to a very low point; but in Mexico this maximum production will doubtless be reached much more slowly, because of: (1) internal political difficulties; (2) delay in opening up isolated districts by railroads and pipe lines; (3) the many large tracts of land in most districts, the expense of handling which keeps out small operators, who would over-drill; (4) the high expense of operating, which keeps out a host of adventurers; (5) the lack of tankers for transportation; (6) the difficulty, by reason of expensive production and transportation, of competing in the United States market with Gulf Coast and California fuel oils; and (7) the present over-production of high-grade oil in the United States, which will delay the adaptation to Mexican oils of the new processes for refining heavy oils. These economic conditions account for the fact that Mexico, from wells having a present daily capacity of from 330,000 to 500,000 bbl., is producing only 70,000 bbl. a day, and that no active endeavors adequate to this discrepancy are now in progress, for the liberation and conveyance to the sea-coast, of the enormous amount of oil thus shut in.



DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

The Cost of Maintaining Production in California Oil Fields

BY M. E. LOMBARDI, BERKELEY, CAL.

(San Francisco Meeting, September, 1915)

THE cost of maintaining the production of an operating oil company is one of the most important, as well as one of the most difficult to estimate, of the various items which go to make up the total cost of producing oil. In the opinion of many operators the cost of drilling and developing new wells, whose production takes the place of the loss in production through decreased yield of the old wells, is considered an operating expense, for the reason that the total income-producing power of the property is not increased. The cost of these new wells may be added directly to production expense, or it may be charged to capital account, and this account depreciated to the same extent. In either case, after an oil company is on a satisfactory income-producing basis, the money for drilling new wells to maintain production must be taken from the income of the company.

Three factors determine the cost of maintaining production: First, the rate of decline in yield of the wells; second, the cost of drilling and equipping new wells in terms of their production; third, the decrease in initial production of new wells as the property is drilled and gas pressure taken off the sands, etc.

In order to check theoretical figures for the cost of maintaining production the writer, with the assistance of Reed Bush and Fred Tough, has compiled the following figures. It is necessary first of all to choose a period of time during which production for the field or territory studied remains fairly constant. The longer the period of time over which this condition prevails, the better. It is also necessary to eliminate or allow for any extraneous condition which may have affected the production returns, such as the shutting down of a large number of wells, the bringing in of a well of exceptionally large yield, the opening up of virgin territory where the flush yield is greater than will be had from new wells in after years, etc.

All these conditions are exemplified most simply in the case of the Coalinga field, and perhaps these figures are more truly prophetic than the two other cases given. In applying these figures to the future it is well to assume that the average yield of new wells will constantly de-

crease, since the flush-production period is past for most of the territory. On the other hand, the cost of drilling will be constantly less, owing to improved methods and particularly to the increasing use of the rotary drill. Although a great many estimations must be made, the results check closely enough to be interesting and possibly of value.

The data used are from the records of the Producers' Agency, the *Standard Oil Bulletin*, and various private sources.

It is to be noted that the figures for a number of wells at the beginning of the periods include producing wells only, but that all wells drilled during the period, whether profitable or not, are included in the calculations, provided they are within the accepted limits of the field—not pure wild cats.

Coalinga Field

In the Coalinga field the period from Aug. 1, 1912 to July 31, 1913 was chosen. The production curve is shown in Fig. 1. It will be noted that

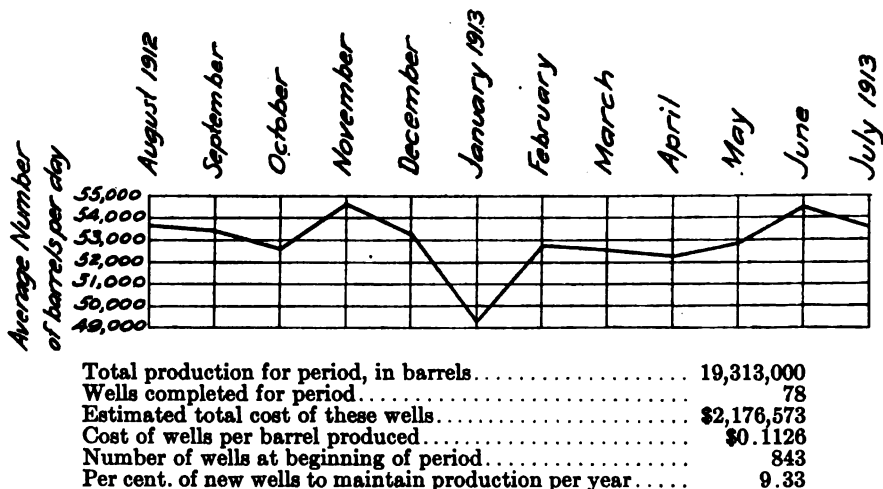


FIG. 1.—CURVE OF TOTAL PRODUCTION OF CRUDE OIL IN COALINGA FIELD, CALIFORNIA, FOR THE PERIOD AUGUST, 1912, TO AUGUST, 1913—12 MONTHS.

the production during these 12 months was remarkably uniform, the greatest variation from the average being about 8 per cent. and the average variation on a monthly basis being about 2.8 per cent. During this period 19,313,000 bbl. of oil were produced and 78 new wells were completed. The number of wells suspended during the period varied from 176 to 214 out of a total of 843 wells at the beginning. These wells were "off" from natural causes, and were not to any extent intentionally shut in. The greatest variation from the average number of wells suspended was only 8.6 per cent., so that this factor of wells shut in is not of great importance in the results.

The main facts are, that at the beginning of this period 843 wells were producing, and that, in order to maintain the production of these wells, 78 new wells had to be drilled. To arrive at the cost of these 78 wells, representing the cost of maintaining production, the new wells were classified in four groups. The actual cost of 95 wells in this field (also divided into groups of similar characteristics) was accurately known to the writer. These average cost figures applied to the 78 wells in question showed their cost to be approximately \$2,176,573. This cost includes derrick and rig, well, flow-tanks and flumes, gas engine or other local pumping equipment, but not such items as storage facilities, pipe lines, main power plants, etc. If this figure is divided by the total number of barrels produced during the period, we find that 11.26 c. per barrel of oil produced is the cost of maintaining production in this field.

It is also interesting to note that the number of wells necessary to maintain production for one year was in this case 9.23 per cent. of the total number of wells producing at the beginning of the period.

Sunset-Midway Field

In the Sunset-Midway field the period from November, 1913 through November, 1914, was chosen. Fig. 2 shows that the production was

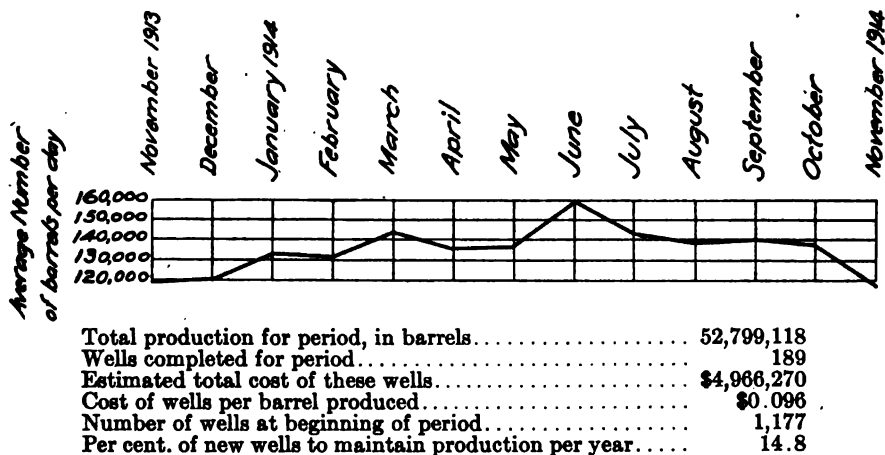


FIG. 2.—CURVE OF TOTAL PRODUCTION OF CRUDE OIL IN SUNSET-MIDWAY FIELD, CALIFORNIA, FOR THE PERIOD NOVEMBER, 1913, TO DECEMBER, 1914—13 MONTHS.

more than maintained, that is, that it was higher at all times during the period than at either end. The Lakeview No. 2 gusher came in in May, 1914 with a production of about 25,000 bbl. However, the greatest variation of the average production was not more than 10.5 per cent. During this period seven gas wells were drilled. These are not included in the figures given, as it is assumed that they were drilled to produce gas only, and no attempt made to produce oil.

During this period, a wide-spread agitation occurred in favor of limiting production, and a good many wells were closed in. The probable production of these closed-in wells was estimated as closely as possible and added to the actual production of the field, in order that this intentional decrease should not affect the results, which are of necessity based on normal flow from each well.

It will be noted that the total oil produced was 52,799,118 bbl. including estimated shut-in production, and that 189 wells were drilled during the period. The cost of these 189 new wells was estimated in the same manner as for the Coalinga field. The costs of 95 wells in various parts of the Sunset-Midway field, being accurately known to the writer, were applied to the 189 wells with due regard to the location of each well, etc., the same items being included as in the case of Coalinga. The resulting estimate is \$4,966,270 for maintaining production during the period, which amounts to 9.6 c. per barrel of oil produced.

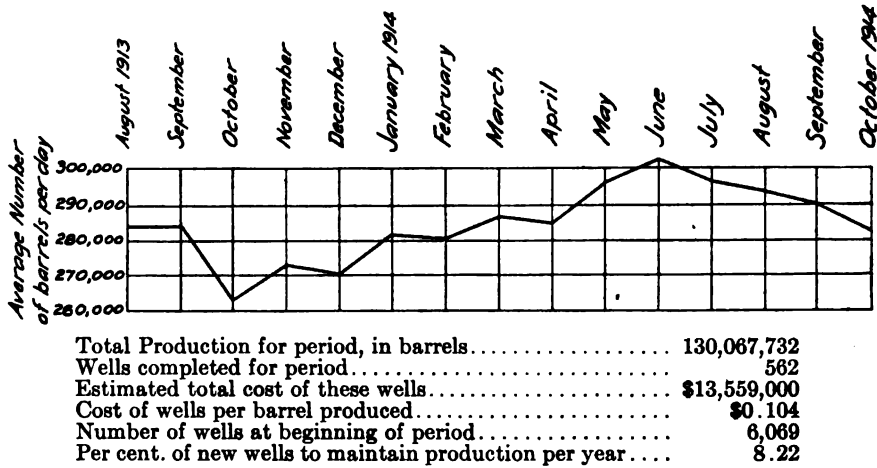


FIG. 3.—CURVE OF TOTAL PRODUCTION OF CRUDE OIL IN CALIFORNIA, FOR THE PERIOD AUGUST, 1913, TO NOVEMBER, 1914—15 MONTHS.

The proportion of new wells necessary to maintain production for one year in this case is 14.8 per cent., which is a considerably higher figure than for Coalinga, although the cost per barrel produced is less. This is explained to some extent by the fact that a larger number of new wells in the Sunset-Midway field were drilled in shallow and “easy” territory in the northwestern part of the field, with a correspondingly low cost per well.

All California Fields

A study was next made based on the production of the whole State of California for a period of 15 months beginning August, 1913. During this period, as shown in Fig. 3, the greatest variation from the average

production was only 7.5 per cent., which is remarkably small considering the period.

At the beginning, approximately 6,069 wells were producing and during the period approximately 562 wells were brought in, costing approximately \$13,559,000. This figure for the cost of new wells is not as accurate as desired, because the writer is not familiar with the cost of wells in some of the smaller fields. Average cost figures taken from over 250 wells in four fields were used. During the period approximately 130,067,730 bbl. of oil were produced, and dividing the cost of the new wells by this production, we find that the cost of maintaining production was 10.4 c. per barrel of oil produced. This differs from the Coalinga figure by only 0.86 c. per barrel, and from the Sunset-Midway figure by 0.8 c. and is very close to a mean between the two. It is also interesting to note that the number of wells necessary to maintain production over the whole State for one year was 8.22 per cent. of the wells producing at the beginning of the year, which compares favorably with the figure of 9.23 per cent. for Coalinga.

Cost per Barrel of Daily Production

It is also interesting to apply the data given above to estimate the cost of maintaining production in terms of the average daily production. For example: A company operating average leases in various fields, having a daily production of several thousand barrels, may wish to estimate from a general average how much money will have to be appropriated per year for new drilling to maintain its production.

In the following calculations the average daily production for the whole period is assumed as the production at the beginning of the period. In Coalinga the daily production may be thus assumed as 52,912 bbl. at the beginning of the period illustrated above. It, therefore, cost \$40.38 per barrel of daily production to maintain production during the ensuing year.

Likewise for the Sunset-Midway field, the figure is \$33.70, and, for all fields in California, \$38.

It must be borne in mind that, in this case, as in the case of the cost per barrel produced, these figures show the cost for wells and their immediate appurtenances only: The cost of extension of pipe lines, new permanent power plants, camps, water systems, etc., must be added. A fair figure for these is between 12 and 20 per cent. of the cost of wells.

Depreciation

If we apply the cost figures used above to all wells producing at the beginning of the various periods shown, the apparent depreciation in earning power of investment during the ensuing year is shown by the

ratio of new wells to old. In Coalinga it is 9.23 per cent., in the Sunset-Midway field 14.8 per cent., and for all California fields 8.22 per cent.

It must be borne in mind that these figures apply only when considering wells producing at the beginning of the period. That is, no account in investment is taken of past failures or abandoned wells. Also that the figures show an average depreciation for wells in all stages of life. It is well known that new wells fall off in production as much as 50 per cent. in their first year, whereas wells 5 or 6 years old decrease very slightly in comparison.

It must also be borne in mind that many of the new wells which maintained the production during the period illustrated were in practically virgin territory in the extension of the known fields, or in new fields, and that their initial production was correspondingly high. This condition will not exist long, and it will be safe to discount the results given above for future use.

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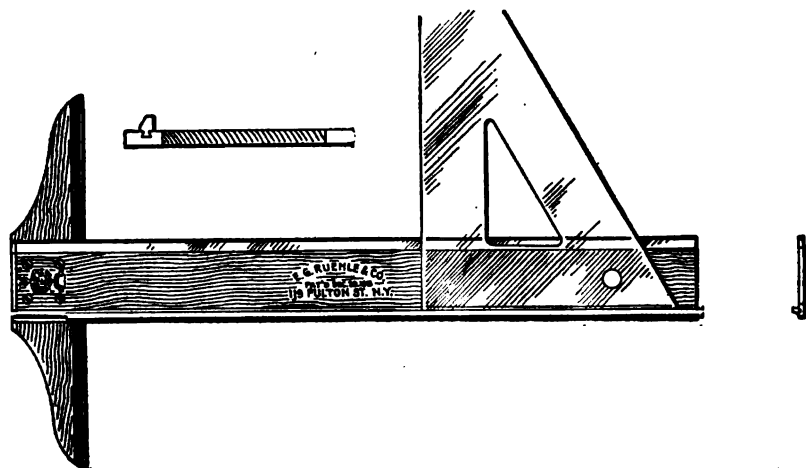
THE AMERICAN INSTITUTE OF MINING ENGINEERS has adopted no official badge, the possession of which is proof of membership. Numerous designs, however, have been executed for temporary use, and of these three here shown have proved so convenient and attractive, that many members continue to wear them as personal ornaments. To this there is no objection; and these badges will be sent postpaid to Members and Associates on the receipt of the price by New York check, or Postal Money Order, in favor of the Institute, as follows:

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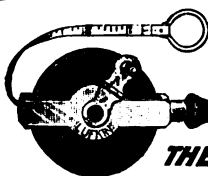
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This slow-speed member, to which is secured the slow-speed shaft, is mounted on two heavy-duty S. K. F. ball bearings, one on each side of the gears, and supported directly by the heavy housing. Thus it will be seen that the slow-speed member and shaft carrying the intermediate gears, and the high-speed shaft and pinion, are independent of each other for support and that each is supported directly by the housing.

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illustration of the operation of their boiler-tube cleaners, automatic cut-off valves, multiple strainers, and grease extractors.

Referring to the illustration of the Exhibit, to the left are the high-power direct motor-driven tube cleaners for boilers, condensers and economizers, suspended and inserted in condenser tubes and the tubes of a B. & W. manifold, just as they are in actual use. Directly behind these cleaners can be seen the handwheels of the Types A and B outside and internal dashpot automatic cut-off valves, and the Type C or internal dashpot non-return valve.

At the back of the exhibit and to the extreme left is an exhibit of the Lagonda Stirling boiler-cleaning device, by which the entire operation of cleaning is conducted from without the drum. The device is supported in an oval frame, representing the manhole, and has the cleaner and rubber operating hose inserted, just as in operation. Immediately to the right of this and to the left of the booth are shown the electric and air or steam motor-driven reseating machines, for reseating and grinding fitting surfaces and cleaning scale and soot from caps and nuts of B. & W. and similar type boilers.



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- ¶ Sometimes it's the fault of the hoist.
- ¶ But isn't it often the fault of *contributing* conditions, or the operator.
- ¶ Take VULCAN hoists, for instance. We take the utmost pains in the making. We test and analyze every casting, every forging; our own specifications are more rigid than yours. There's not a thing left undone to make maintenance *low*.
- ¶ And if it is not low, it's generally traceable to outside conditions that can be remedied easily.
- ¶ Keep your hoist clean, well-oiled and greased and test all the nuts and bolts now and then, and give the bearing wedges a look.
- ¶ Then you'll see maintenance worries disappear.

VULCAN IRON WORKS
1744 Main Street
Wilkes-Barre, Pa.

The demonstration of the working of the multiple strainer is especially interesting. A three-basket type strainer has been constructed with a glass panel at one end, so that the flow of water through the strainer and the accumulation of solid matter in the basket can be seen. In operation the small two-basket strainer body shown at the left is used for inserting sticks, leaves and other such refuse as is found in plant-supply water, then the water is turned on and allowed to flow through the strainer. All the solid matter passing into the strainer is caught in the strainer baskets. It is then demonstrated how the strainer baskets can be individually drawn out of commission, removed and cleaned without interrupting the operation of the other baskets or the flow of water.

Charles C. Moore & Co., Western representatives of the Lagonda Company, have charge of the exhibit.

New Type of Electric Power Drill

The accompanying illustration shows a new type of powerful coal drill recently developed and placed on the market by the Jeffrey Manufacturing Co., and known as their A-5-C electric power drill.



“ELECTRIC-AIR” DRILLS

For the small or isolated mine or contract where electric current is available this type of drill performs a special mission.

The electric-air drill combines with the flexibility of the electric current as power the efficiency of the compressed air drill in a complete power plant unit of great compactness and portability, solving the power plant problem and eliminating the expense of “in the mine” piping.

It is durably constructed throughout and compares favorably in drilling capacity with the standard piston drills of equal size.

It is built in three sizes.

Bulletin No. 4209

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THE ENSIGN BICKFORD CO., Simsbury, Conn.
Original Manufacturers of Safety Fuse

Established 1836

The motor and drill mechanism proper are mounted on trunnions in a carriage, which may be raised or lowered in a vertical channel-iron frame by means of a $\frac{3}{8}$ -in. steel rope. One end of this rope is fastened to the top yoke of the frame and the other end to a drum on the carriage. This drum is rotated by a handle, as shown, to wind up the rope and to raise the carriage. A ratchet and pawl are provided on the drum to prevent the carriage from dropping when the handle is released.

When it is desired to move the drill from one place to another, the carriage carrying the motor and mechanism is lowered to the bottom of the frame and the drill rolled along on the wheels provided on the bottom cross yoke, in a manner similar to an ordinary hand freight truck.

The motor of this drill is rated at 3 hp. The total weight of the drill is approximately 300 lb., while it is made sufficiently rugged and powerful to drill through any material against which the auger will stand up.

With the means employed on this drill for raising and lowering and with the wheels provided for moving it from place to place, only one man is required to move, set up, adjust and operate the machine.

Further particulars concerning this drill, also latest catalogues and bulletins on the Jeffrey complete line of coal-mine and tippie machinery, may be obtained by writing to the Jeffrey Manufacturing Co., 902 North Fourth Street, Columbus, Ohio, or to any of their branch offices.

How to Keep Conveyor-Belt Records

A new method of recording the cost of various conveyor belts in operation has been worked out by the B. F. Goodrich Co., of Akron, Ohio, in the interest of all belt users.

Ordinarily accurate costs are rather difficult to determine and many operators have not the time to work out an adequate system for making computations. The use of various belts, however, is an important factor and many a dollar can be saved when you know where to cut down the tonnage cost.

The B. F. Goodrich Co. announces that it will send at its expense a card that is simple, but most efficient, which will help belt users.



Largest Motor Driven Hoist in the United States
The North Butte Mining Co., Butte, Mont.

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is operated by a Westinghouse motor with a Westinghouse flywheel-motor-generator set.

Absolute reliability was the first requirement for this hoist and Westinghouse apparatus was selected for that reason.

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Net load of material hoisted	7 to 8 tons
Weight of Skip	9000 lbs.
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Output of mine in one hour	320 tons from 2000 ft. level
Output of mine in one hour	250 tons from 3000 ft. level
Output of mine in one hour	200 tons from 4000 ft. level
Maximum Hoisting Speed	3000 ft. per minute
Normal Operation	Skips Balanced
Emergency Operation	Skip Unbalanced

Our engineers have been able to design and install apparatus for the successful operation of this largest electric hoist. Let them show you how to hoist at minimum cost.

What we have done for them we can do for you. Send for booklet "Electric Drive for Mine Hoists," No. 3113.

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WESTON ELECTRICAL INSTRUMENT CO. Waverly Park, Newark, N. J. Weston Eclipse AMMETERS, MILLIAMMETERS and VOLTMETERS are well suited for D. C. Circuits of small mine plants.	AMMETERS AND VOLTMETERS
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ALPHABETICAL LIST OF ADVERTISERS

	PAGE
Albany Lubricating Co.	5
Allis-Chalmers Manufacturing Co.	16
Beer, Sondheimer & Co.	21
A. S. Cameron Steam Pump Works	11
Crocker-Wheeler Co.	16
Denver Fire Clay Co.	16
Denver Rock Drill Co.	16
Dorr Cyanide Machinery Co.	5
Edison Storage Battery Co.	Inside Back Cover
Ensign Bickford Co..	13
General Electric Co..	Outside Back Cover
Goodman Manufacturing Co.	17
B. F. Goodrich Co.	Inside Front Cover
W. & L. E. Gurley	1
Harbison-Walker Refractories Co.	18
Illinois Zinc Co.	18
Ingersoll-Rand Co.	13
Jeffrey Manufacturing Co.	9
Lead Lined Iron Pipe Co.	18
A. Leschen & Sons Rope Co.	19
Lufkin Rule Co.	5
Mashek Engineering Co.	19
Robins Conveying Belt Co.	19
John A. Roebling's Sons Co.	9
P. H. & F. M. Roots Co.	19
E. G. Ruehle & Co.	1
Chas. A. Schieren.	5
Sullivan Machinery Co.	7
Terry & Tench Co.	20
Traylor Engineering & Manufacturing Co.	7
Union Iron Works	3
L. Vogelstein & Co.	23
Vulcan Iron Works.	11
Westinghouse Electric & Mfg. Co..	15
Weston Electrical Instrument Co.	1



OCTOBER, 1915



Bulletin of the American Institute of Mining Engineers

PUBLISHED MONTHLY

SECTION I

	PAGE		PAGE
Personal.....	iv	Local Section News	xiii
Employment.....	v	Library.....	xiv
Nominations for Officers.....	vi	Membership.....	xvi
National Reserve Corps of Engineers....	vi	Institute Committees.....	xxviii
Advisory Board for United States Navy..	vii	Officers and Directors.....	xxxvi
Standards for Graphic Presentation.....	ix		

SECTION II.—PAPERS AND DISCUSSIONS

Recent Advances in the Chemistry of the Cyanogen Compounds. By J. E. Clennell, B. Sc.	2115
The Control of Chill in Cast Iron. Considering the Elements Effective in the Manufacture of Malleable Castings and Chilled Car Wheels. By Grafton M. Thrasher	2129
The Newnam Hearth. By William E. Newnam, B. S.	2139
Geology of the Ore Deposits of the Tintic Mining District. By Guy W. Crane, B. A.	2147
Measurement of the Temperature Drop in Blast-Furnace Hot-Blast Mains. By R. J. Wysor	2161
The Iron Deposits of Daiquiri, Cuba. By Waldemar Lindgren and Clyde P. Ross	2171
Industrial Section	Interpolated in Advertising Section.

The curve on my tonnage
cost chart has taken a
slant toward the low
figure since we put in
those Goodrich Belts.
Supt. Wise

Uninterrupted service is one factor in the reduction of tonnage costs—by Goodrich belts. No shut-downs—no delays.

Flexibility is another Goodrich tonnage cost-cutter. No spillage and less wear.

"Longlife"—*longer* service—gives you more tons per *initial* cost.

An edge construction that will not tear off is a feature also of our strong, perfectly balanced belts.

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CONVEYOR BELTS

are so skillfully proportioned—so well fitted for the work *you* want done, and so well fortified by duck and friction and rubber cover that you *need* them in your plant. If you want to see your tonnage costs come down "Get in touch with Goodrich."

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Packing
Valves, etc.

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Factories: Akron, Ohio

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(Name in Full)

*Address*_____

Signatures of three
Members or
Associates.

**Signatures of three
Members or
Associates.**

*Education, general and technical, when, where and how acquired,
with degrees, if any.*

[illegible]

[illegible]

*Present position*_____

Signature.

Dated _____ *191* _____

ARTICLE II.—MEMBERS.

SEC. 2. The following classes of persons shall be eligible for membership in the Institute, namely : as Members, all professional mining engineers, geologists, metallurgists, or chemists, and all persons actively engaged in mining and metallurgical engineering, geology, or chemistry ; as Associates, all persons desirous of being connected with the Institute who in the opinion of the Board of Directors are suitable.

Every candidate for election as a Member, Associate, or Junior Member must be proposed for election by at least three Members or Associates, must be approved by the Committee on Membership, as prescribed in the By-Laws, and must be elected by the Board of Directors.

Harvard College Library
Aug. 14, 1916.
Bequest of
Erasmus Darwin Leavitt.

**BULLETIN OF THE
AMERICAN INSTITUTE OF
MINING ENGINEERS**
PUBLISHED MONTHLY

No. 106

OCTOBER

1915

Published Monthly by the American Institute of Mining Engineers at 212-218 York St., York, Pa., H. A. WISOTZKEY, Publication Manager. Editorial Office, 29 West 39th St., New York, N. Y., BRADLEY STOUTENON, Editor, F. O. PINNEN, Asst. Editor. Cable address, "Aime," Western Union Telegraph Code. Subscription (including postage), \$10 per annum; to members of the Institute, public libraries, educational institutions and technical societies, \$5 per annum. Single copies (including postage), \$1 each; to members of the Institute, public libraries, etc., 50 cents each.

Entered as Second Class matter January 23, 1914, at the Post Office at
York, Pennsylvania, under the Act of March 3, 1879.

MANUSCRIPTS FOR THE FEBRUARY MEETING

The 112th meeting of the Institute for the presentation and discussion of technical papers, and the Annual Business Meeting, will be held at the Headquarters of the Institute in New York City, Feb. 14 to 17, 1916.

All papers to be presented at this meeting of the Institute must be in the hands of the Secretary on or before Dec. 1, 1915.

PERSONAL

(Members are urged to send in for this column any notes of interest concerning themselves or their fellow-members.)

Members and guests who registered at Institute headquarters during the period Aug. 10 to Sept. 10, 1915:

Everett L. Booth, New York, N. Y.

T. E. Sturtevant, Dover, N. J.

C. A. Filteau, New York, N. Y.

Guy C. Riddell, Helena, Mont.

Joseph Leiner, New York, N. Y.

Alfred W. G. Wilson, Ottawa, Canada.

Raymond B. Ladoo, Cambridge, Mass.

Henry A. Tobelman, Ajo, Ariz.

R. B. Tinsley, Guanoco, Venezuela.

E. H. Hamilton, Norfolk, Va.

M. Yoshimura, Tokyo, Japan.

Van H. Manning has been appointed Director of the U. S. Bureau of Mines, to succeed the late Dr. Joseph A. Holmes. Mr. Manning was Assistant Director of the Bureau.

O. M. Lewyn, formerly Mine Superintendent for the Braden Copper Co., has accepted the position of Assistant Mine Manager with the Union Minière de Haut Katanga of South Africa.

Royal P. Jarvis has been appointed Chief Chemist for the Cananea Consolidated Copper Co., Cananea, Sonora, Mexico.

Robert M. Keeney has taken a position with the Standard Chemical Co., Canonsburg, Pa.

L. V. Emanuel is now with the River Smelting & Refining Co., Florence, Colo.

Hugh D. Pallister, formerly Assistant Professor of Mining at the Pennsylvania State College, has resigned to accept the position of Professor of Geology and Mining at the State School of Mines of the University of Texas, El Paso.

H. H. Utley has accepted the position of Manager of the River Smelting & Refining Co., Florence, Colo.

R. O. Hocking has accepted a position as Inspector of Mining Operations with the Meriden Iron Co., Hibbing, Minn.

F. C. Newton is Vice-President and General Manager of the Newton Manufacturing Co., Lowell, Mass.

Adolphe E. Borie, Chairman of the Board of Directors of the American Road Machinery Co., has been elected president of the Driggs-Seabury Ordnance Co., of Sharon, Pa.

E. S. Dickinson, formerly Assistant Professor of Mining at the University of Kansas, has accepted a position on the mining staff of the Canadian Copper Co., Copper Cliff, Ont., Canada.

J. L. W. Birkinbine has been appointed by the Secretary of the Interior as consulting engineer of the Bureau of Mines. Mr. Birkinbine will prepare a monograph on the manufacture of iron and steel in this country.

Andrew C. Lawson, Professor of Geology, has been appointed Dean of the College of Mining, University of California.

Walter S. Weeks has been appointed Associate Professor of Mining, University of California.

Robert Rhea Goodrich has been appointed Professor of Metallurgy at the University of Idaho, Moscow, Ida.

James F. McCarthy has been appointed Manager of the National Copper Mining Co., Mullan, Ida.

POSITIONS VACANT

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons.)

Superintendent for iron mine operated on the sub-level and caving system. Small output, moderate salary. Location, Southern State. No. 45.

Chemist to sample and analyze iron ore. Experience in other branches not essential. Location, Southern State. No. 46.

ENGINEERS AVAILABLE

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons introduced by members.)

Experienced mine superintendent desires position. Several years' experience in practical management of mining and development work. Has also practiced as consulting engineer and is thoroughly experienced in the examination and development of mining properties. Mexican experience. Will go anywhere for reliable concern. Salary moderate. References. No. 254.

Member, mechanical superintendent, wants position. Has had 12 years' experience as master mechanic of smelting plants in United States and Mexico. Also superintendent of shops doing marine, mining, and general contracting work. Experienced in manufacture of mining tools and equipment. Up to the minute in copper, lead, and zinc smelting. Has learned a great deal, can forget much, and has the inclination and ability to learn more. Best of references. No. 255.

Member, technical graduate, aged 25, unmarried, three years' experience in metal mining and geology, Arizona. Available immediately as assistant to consulting engineer in mine examination or in mine operation. Interview in New York. Best of references. No. 256.

Member, technical graduate, aged 31, unmarried. Thoroughly experienced in operative and executive departments of bituminous coal mining. Also considerable experience in railroad, hydro-electric and general construction work. Will go anywhere, but prefers the West or South. At present temporarily employed on Pacific Coast. Can furnish best of references. No. 257.

Member, graduate engineer, aged 44, with over 22 years' experience in connection with the development of large coal and coke properties in different parts of the country, desires position as general manager or general superintendent of large bituminous coal producing, or will take an interest and the management of a smaller operation. No. 258.

NOMINATIONS FOR OFFICERS

The co-operation of the members of the Institute is earnestly sought by the Committee on Nominations, recently appointed by the Board of Directors, in its work of formulating a ticket for officers and places falling vacant in the Board of Directors in February, 1916. The Committee must present the official ticket of nominations to the Board of Directors at its October meeting. Members are, therefore, urged to send promptly their nominations for officers and Directors in order that the Committee may have an opportunity to give them full consideration.

The officers to be elected at the annual meeting in February, 1916, are: One officer, known as Director and President. Two officers, known as Director and Vice-President. Five officers, known as Director.

The officers of the Institute whose terms expire are as follows:

President, William L. Saunders (not eligible for re-election).

Past President, Charles F. Rand.

Vice-Presidents, Thomas H. Leggett, District 0; Fred W. Denton, District 4.

Directors, John W. Finch, District 7; John H. Janeway, District 0; Edward P. Mathewson, District 5; Joseph W. Richards, District 2; George D. Barron, District 0.

Communications should be sent to the Chairman of the Committee on Nominations, Crocker Bldg., San Francisco, Cal.

COMMITTEE ON NOMINATIONS:

F. W. BRADLEY, *Chairman*.

LOUIS S. CATES,
R. C. GEMMELL,

JAMES F. KEMP,
FRANK A. ROSS,

FRANK M. SMITH,
ARTHUR THACHER.

NATIONAL RESERVE CORPS OF ENGINEERS

Last spring the suggestion was made that the national engineering societies offer to assist the War Department in the formation of an engineer reserve in the United States Army. The suggestion was brought to the attention of some of the officers of the Army, who commended it and confirmed the belief that the need of such a reserve was a real one; that the European war had distinctly shown that engineers in time of war are required on a scale never before realized and of great varieties of experience. In our own army no adequate provision is made for great numbers of engineers, for although the corps of engineers is composed of men of the very highest professional proficiency in certain lines it is greatly short of the numerical strength that would be immediately required.

Acting on this suggestion, the American Society of Civil Engineers, the American Institute of Mining Engineers, the American Society of Mechanical Engineers, the American Institute of Electrical Engineers and the American Institute of Consulting Engineers appointed committees authorized and directed to take such steps as might be advisable to urge the organization of such a reserve corps as part of the regular army, and so assist in putting the United States in a better condition of prepared-

ness against war. Conferences were had with the Secretary of War and the officers of the General Staff and of the War College. So much encouragement was received that it was felt advisable to form a single committee to represent jointly the five societies, in short, the profession as a whole. This will simplify future conferences with those in authority and show that the various engineering organizations are earnestly co-operating.

Accordingly, a joint committee has now been formed, after obtaining formal permission from the five societies, consisting of the chairmen of the previously appointed society committees, as follows:

WILLIAM BARCLAY PARSONS, Chairman Committee,
American Society of Civil Engineers,
HENRY S. DRINKER, Chairman Committee,
American Institute of Mining Engineers,
WILLIAM H. WILEY, Chairman Committee,
American Society of Mechanical Engineers,
BION J. ARNOLD, Chairman Committee,
American Institute of Electrical Engineers,
RALPH D. MERSHON, Chairman Committee,
American Institute of Consulting Engineers.

This committee has selected Mr. Parsons as its Chairman and is in communication with the War Department.

Before such a reserve corps can be formed, legislation authorizing it must be passed by Congress and approved by the President. The details of such legislation are now being studied by the War Department. While no decision has been reached, it is under consideration to issue commissions as officers to such engineers as will meet certain professional and physical standards, and who in times of war would be subject to orders from the Secretary of War as other officers of the army, and in times of peace would perform such duty as would not seriously interfere with ordinary work, but give each officer some military education and experience. In this way it is hoped to have a large body of engineers which could be called quickly on duty.

The separate committees are still in existence to do the necessary work among their fellow members of the several societies as soon as the decision of the War Department is given, and a general scheme of organization adopted.

WILLIAM BARCLAY PARSONS, *Chairman*,
HENRY S. DRINKER,
WILLIAM H. WILEY,
BION J. ARNOLD,
RALPH D. MERSHON,

Joint Committee of the National Engineering Societies on the National Reserve Corps of Engineers.

The members of the Committee of the American Institute of Mining Engineers on the Engineers Reserve Corps movement are: Henry Sturgis Drinker, *Chairman*; Arthur S. Dwight, Warren A. Wilbur.

NAVAL ADVISORY BOARD ON INVENTIONS

The personnel of the Naval Advisory Board on Inventions, of which Thomas A. Edison is to be Chairman, has been announced by Secretary of the Navy Daniels, as follows:

W. R. WHITNEY	}	American Chemical Society.
L. H. BAEKELAND		
FRANK JULIAN SPRAGUE	}	American Institute of Electrical Engineers.
B. G. LAMME		
ROBERT SIMPSON WOODWARD	}	American Mathematical Society.
ARTHUR GORDON WEBSTER		
ANDREW MURRAY HUNT	}	American Society of Civil Engineers.
ALFRED CRAVEN		
MATTHEW BACON SELLERS	}	American Aeronautical Society.
HUDSON MAXIM		
PETER COOPER HEWITT	}	Inventors' Guild.
THOMAS ROBINS		
HOWARD E. COFFIN	}	American Society of Automobile Engineers.
ANDREW L. RIKER		
WILLIAM LAWRENCE SAUNDERS	}	American Institute of Mining Engineers.
BENJAMIN BOWDITCH THAYER		
JOSEPH WILLIAM RICHARDS	}	American Electrochemical Society.
LAWRENCE ADDICKS		
WILLIAM LE ROY EMMET	}	American Society of Mechanical Engineers.
SPENCER MILLER		
HENRY ALEXANDER WISE WOOD	}	American Society of Aeronautic Engineers.
ELMER AMBROSE SPERRY		

In announcing the names of the members of the board, Secretary Daniels made the following statement:

"Desiring to make available the latent inventive genius of our country to improve our navy, a short while ago I requested Mr. Thomas A. Edison to become Chairman of an Advisory Board of eminent men who would make up the board. Mr. Edison, with the patriotism characteristic of American inventors, accepted the call to duty. The plan adopted for selecting the members of the Advisory Board was as follows:

"I requested eleven great engineering and scientific societies to select by popular election two members each to represent their society on the board. The result has been most gratifying. I have received the nominations of all these societies and have accepted them, and it only remains to have a meeting, organize and determine the method of procedure in order to utilize to the best advantage of our navy this mobilization of the talent and genius of our great country. The responses of the various societies invited to co-operate in the great undertaking indicate the patriotic enthusiasm awakened by this call to duty."

AFFILIATED STUDENT SOCIETY NOTES

The Mining Association of the University of California, Berkeley, Cal., has elected officers for the fall term, as follows:

President, O. Cavins,
Vice-President, W. Arendt,
Treasurer, C. N. Schuette,
Alumni Secretary, E. Woodcock,
Secretary, L. J. Brunel.

Executive Committee

E. Y. Dougherty

W. Farnlacher

W. Miller

JOINT COMMITTEE ON STANDARDS FOR GRAPHIC PRESENTATION

PRELIMINARY REPORT PUBLISHED FOR THE PURPOSE OF INVITING SUGGESTIONS FOR THE BENEFIT OF THE COMMITTEE¹

As a result of invitations extended by The American Society of Mechanical Engineers, a number of associations of national scope have appointed representatives on a Joint Committee on Standards for Graphic Presentation. Below are the names of the members of the committee and of the associations which have co-operated in its formation.

WILLARD C. BRINTON, *Chairman*, American Society of Mechanical Engineers.

[7 East 42nd Street,

New York City.]

LEONARD P. AYRES, *Secretary*, American Statistical Association.

[130 East 22nd Street,

New York City.]

N. A. CARLE, American Institute of Electrical Engineers.

ROBERT E. CHADDOCK, American Association for the Advancement of Science.

FREDERICK A. CLEVELAND, American Academy of Political and Social Science.

H. E. CRAMPTON, American Genetic Association.

WALTER S. GIFFORD, American Economic Association.

J. ARTHUR HARRIS, American Society of Naturalists.

H. E. HAWKES, American Mathematical Society.

JOSEPH A. HILL, United States Census Bureau.

HENRY D. HUBBARD, United States Bureau of Standards.

ROBERT H. MONTGOMERY, American Association of Public Accountants.

HENRY H. NORRIS, Society for the Promotion of Engineering Education.

ALEXANDER SMITH, American Chemical Society.

JUDD STEWART, American Institute of Mining Engineers.

WENDALL M. STRONG, Actuarial Society of America.

EDWARD L. THORNDIKE, American Psychological Association.

The committee is making a study of the methods used in different fields of endeavor for presenting statistical and quantitative data in graphic form. As civilization advances there is being brought to the attention of the average individual a constantly increasing volume of comparative figures and general data of a scientific, technical and statistical nature. The graphic method permits the presentation of such figures and data with a great saving of time and also with more clearness than would otherwise be obtained. If simple and convenient standards can be found and made generally known, there will be possible a more universal use of graphic methods with a consequent gain to mankind because of the greater speed and accuracy with which complex information may be imparted and interpreted.

The Following are Suggestions which the Committee has thus far Considered as Representing the more Generally Applicable Principles of Elementary Graphic Presentation

1. The general arrangement of a diagram should proceed from left to right.

¹ Copies may be had from the office of the Institute, 29 West 39th St., New York 10 cents each. Discount in quantities.

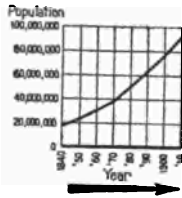


FIG. 1.

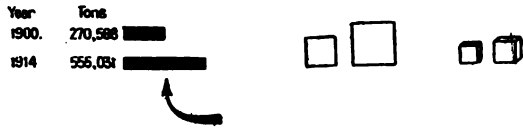


FIG. 2.

2. Where possible represent quantities by linear magnitudes as areas or volumes are more likely to be misinterpreted.



FIG. 3.

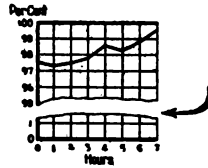


FIG. 4.

3. For a curve the vertical scale, whenever practicable, should be so selected that the zero line will appear on the diagram.

4. If the zero line of the vertical scale will not normally appear on the curve diagram, the zero line should be shown by the use of a horizontal break in the diagram.

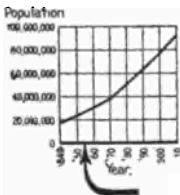


FIG. 5A.

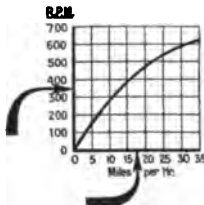


FIG. 5B.

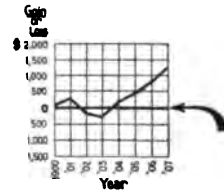


FIG. 5C.

5. The zero lines of the scales for a curve should be sharply distinguished from the other co-ordinate lines.

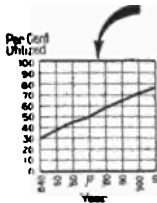


FIG. 6A.

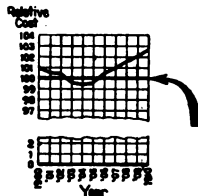


FIG. 6B.

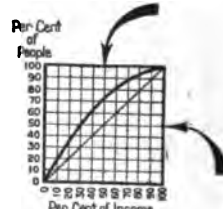


FIG. 6C.

6. For curves having a scale representing percentages, it is usually desirable to emphasize in some distinctive way the 100 per cent. line or other line used as a basis of comparison.

7. When the scale of a diagram refers to dates, and the period represented is not a complete unit, it is better not to emphasize the first and last ordinates, since such a diagram does not represent the beginning or end of time.

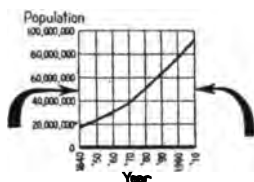


FIG. 7.

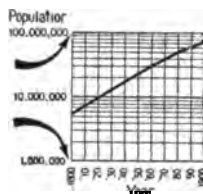


FIG. 8.

8. When curves are drawn on logarithmic co-ordinates, the limiting lines of the diagram should each be at some power of ten on the logarithmic scales.

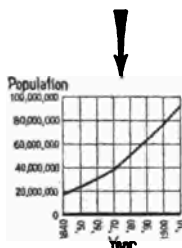


FIG. 9A.

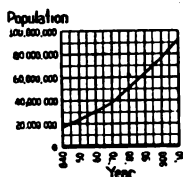


FIG. 9B.

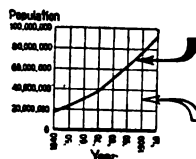


FIG. 10.

9. It is advisable not to show any more co-ordinate lines than necessary to guide the eye in reading the diagram.

10. The curve lines of a diagram should be sharply distinguished from the ruling.

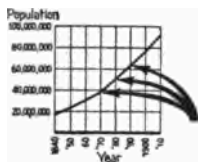


FIG. 11A.

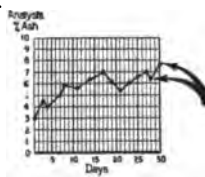


FIG. 11B.

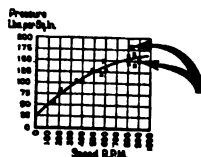


FIG. 11C.

11. In curves representing a series of observations, it is advisable, whenever possible, to indicate clearly on the diagram all the points representing the separate observations.

12. The horizontal scale for curves should usually read from left to right and the vertical scale from bottom to top.

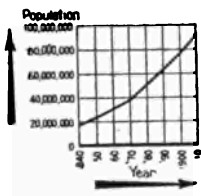


FIG. 12.

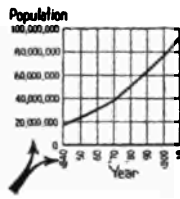


FIG. 13A.

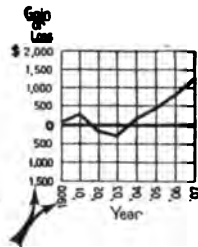


FIG. 13B.

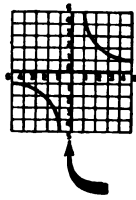


FIG. 13C.

13. Figures for the scales of a diagram should be placed at the left and at the bottom or along the respective axes.

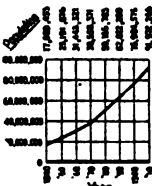


FIG. 14A.

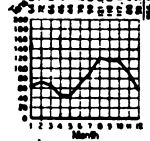


FIG. 14B.

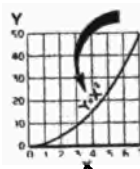


FIG. 14C.

14. It is often desirable to include in the diagram the numerical data or formulæ represented.

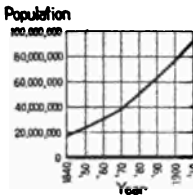


FIG. 15.

Year	Population
1840	17,069,453
1850	23,191,876
1860	31,443,321
1870	38,558,371
1880	50,155,783
1890	62,622,250
1900	75,994,575
1910	91,972,266

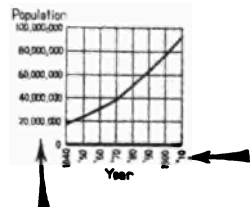
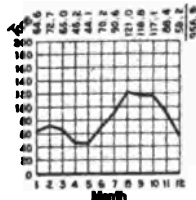


FIG. 16.

15. If numerical data are not included in the diagram it is desirable to give the data in tabular form accompanying the diagram.

16. All lettering and all figures on a diagram should be placed so as to be easily read from the base as the bottom, or from the right-hand edge of the diagram as the bottom.



ALUMINUM CASTINGS OUTPUT OF PLANT No. 2, BY MONTHS. 1914 OUTPUT IS GIVEN IN SHORT TONS; SALES OF SCRAP ALUMINUM ARE NOT INCLUDED.

FIG. 17.

17. The title of a diagram should be made as clear and complete as possible. Sub-titles or descriptions should be added if necessary to insure clearness.

SOUTHERN CALIFORNIA LOCAL SECTION*Executive Committee*SEELEY W. MUDD, *Chairman*C. COLCOCK JONES, *Vice-Chairman*FREDERICK J. H. MERRILL, *Secretary-Treasurer*, 631 Higgins Bldg.,
Los Angeles, Cal.

RALPH ARNOLD

A. B. CARPENTER

In order to make the meetings of the Section more interesting and fruitful, the Executive Committee has formed a sub-committee on programs for Section meetings, which announces for the coming autumn and winter season, four dinner meetings, at intervals of two months, in which the co-operation of all Section members is urgently requested.

The subjects chosen for discussion at these meetings, with the proposed dates, are as follows:

Sept. 28, 1915. Mining Geology of Mojave County, Arizona.

Dec. 14, 1915. General Geology and Mining Conditions in Southern California.

Feb. 1, 1916. Petroleum, its Geology and Technology.

April 4, 1916. General Mining Conditions in South America.

F. J. H. MERRILL, *Secretary*.**PENNSYLVANIA ANTHRACITE SECTION***Executive Committee*R. V. NORRIS, *Chairman*.CHARLES F. HUBER, *Vice-Chairman*, W. J. RICHARDS, *Vice-Chairman*,EDWIN LUDLOW, *Vice-Chairman*, ARTHUR H. STORRS, *Vice-Chairman*.CHARLES ENZIAN, *Secretary-Treasurer*, U. S. Bureau of Mines,
Wilkes-Barre, Pa.DOUGLAS BUNTING,
RUFUS J. FOSTER,FRANK A. HILL,
JOHN M. HUMPHREY,ALBERT B. JESSUP,
ROBERT A. QUIN.

The first annual meeting of the Pennsylvania Anthracite Section was held following a luncheon as guests of the Lehigh Coal & Navigation Co., at the Old Company's Club House, Lansford, Pa., June 11, 1915, Chairman Norris presiding. The luncheon and meeting were attended by 173 members and guests.

The officers and members of the Executive Committee were re-elected for the year 1915.

W. L. Saunders, President of the Institute, in a brief address complimented the Section on the great success attained in membership and contributions to the Institute and the splendid program and entertainment for the members and guests provided by the Lehigh Coal & Navigation Co.

Following President Saunders's remarks the meeting adjourned with a rising vote of thanks and appreciation to the Committee on Arrangements and the management of the Lehigh Coal & Navigation Co.

Visits were made to the Hauto power plant, the Hauto washery, the fire barrier strippings at Summit Hill, and the Greenwood, Rahn, and Lansford collieries of the Lehigh Coal & Navigation Co., and the Bear Creek dam of the Panther Valley Water Co.

CHARLES ENZIAN, *Secretary*.

LIBRARY**AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS****AMERICAN SOCIETY OF MECHANICAL ENGINEERS****AMERICAN INSTITUTE OF MINING ENGINEERS****UNITED ENGINEERING SOCIETY****WILLIAM P. CUTTER, Librarian**

The Library of the above-named Societies is open from 9 A.M. to 10 P.M. on all week-days, except holidays, from September 1 to June 30, and from 9 A.M. to 6 P.M. during July and August. The Library contains about 55,000 volumes, including sets of technical periodicals and the publications of scientific and technical societies.

Members of the Institute, with few exceptions, are forced to spend a portion of their time in localities isolated from sources of information. To these the Library can render valuable service through correspondence; letters requesting information will receive special attention. The Library is prepared to furnish references and copies of articles on mining and metallurgical subjects; to determine the existence of mining maps, and to furnish general information as to the geology and mineral resources of all countries.

All communications should be made as definite as possible so that the information received may be what is desired and not include collateral matter which may not be of interest. The time spent in searching for such collateral matter will be saved, and the information will be sent more promptly and in more usable shape.

The members of the Institute can be of service to the Library by forwarding copies of mining reports, maps privately issued, and similar material, which will be classified, indexed, and made available to other members. Suggestions for additions to the Library, either by purchase or personal solicitation as gifts, will be welcomed. It is hoped that members while in the city will use the Library freely, and assurance is given that most careful service will be rendered to them.

LIBRARY ACCESSIONS**PARTIAL LIST CLASSIFIED BY SUBJECTS****Geology, Mineralogy, and Mineral Resources**

- ANTICLINAL STRUCTURE IN PARTS OF COTTON AND JEFFERSON COUNTIES, OKLAHOMA.** (Bull. 602, U. S. Geological Survey.) Washington, 1915.
- STRUCTURAL RELATIONS OF THE PRE-CAMBRIAN AND PALÆOZOIC ROCKS NORTH OF THE OTTAWA AND ST. LAWRENCE VALLEYS.** (Museum Bull. 18, Canada Mines Department.)
- PRACTICAL OIL GEOLOGY.** By Dorsey Hager. New York, 1915.
- GUIDEBOOK OF THE WESTERN UNITED STATES. Part A. The Northern Pacific Route.** (Bull. 611, U. S. Geological Survey.) Washington, 1915.
- ANALYSES OF ROCKS AND MINERALS FROM THE LABORATORY OF THE UNITED STATES GEOLOGICAL SURVEY, 1880-1914.** (Bull. 591, U. S. Geological Survey.) Washington, 1915.
- LIST OF CANADIAN MINERAL OCCURRENCES.** (Memoir 74, Canada Mines Department.) Ottawa, 1915.

WESTERN AUSTRALIA GEOLOGICAL SURVEY. Annual Progress Report, 1914. Perth, 1915.
BRITISH COLUMBIA MINISTER OF MINES. Annual Report, 1914. Victoria, 1915.
PRODUCTS AND BY-PRODUCTS OF COAL. Ottawa, 1915.

Trade Catalogues

BESSEMER GAS ENGINE Co., Grove City, Pa. Bessemer Monthly. August, 1915.
BROWN HOISTING MACHINERY Co., Cleveland, Ohio.
Catalog U. Brownhoist transfer cars.
Brownhoist Shnoble patent drag-line bucket.
CHICAGO PNEUMATIC TOOL Co., Chicago, Ill. Bull. 130, Lubrication of Pneumatic Tools. August, 1915.
E. I. DU PONT DE NEMOURS POWDER Co., Wilmington, Del.
Du Pont Magazine, August, 1915.
Farmers' Handbook. Instructions in the use of high explosives for clearing land, planting and cultivating trees, etc. Wilmington, 1915.
NATIONAL TRANSIT Co., Oil City, Pa.
Bull. 402. Vertical Gas Engines. Bull. 501. Vertical Oil Engines.
SHORE INSTRUMENT & MANUFACTURING Co., New York, N. Y. Shore Scleroscope for measuring the hardness of metals. 80 pp.
SPARTA IRON WORKS Co., Sparta, Wis. Sludge Bucket. September, 1915.
STEPHENS-ADAMSON MFG. Co., Aurora, Ill. Labor Saver. August, 1915.
WEISS INSTRUMENT Co., Denver, Colo. Description of telescope.

MEMBERSHIP

- NEW MEMBERS

The following list comprises the names of those persons who became members during the period Aug. 10 to Sept. 10, 1915:

Members

- ACOSTA, JOSÉ NICANDRO, Min. Engr. and Chem., Shannon Copper Co.,
Gleeson, Ariz.
- ANDERSON, AMIL ABNER, Met. Homestake Mining Co., Lead, S. D.
- BABB, P. A. Apartado 92, Pachuca, Hid., Mexico.
- BERGH, SVEN VILHELM, Min. Engr. Rorstrandsgatan 32, Stockholm, Sweden.
- BOAS, ROSS H., Min. Engr., Care Highland Boy Mine, Bingham Canyon, Utah.
- BONSALL, JAMES MALCOLM, Chem., Copper Queen Reduction Works, Douglas, Ariz.
- BOWLES, ROSS., Supt. of Smelters, Granby Min. & Smelt. Co., East St. Louis, Ill.
- BRIGHT, GRAHAM, Engr., Westinghouse Electric & Mfg. Co., East Pittsburgh, Pa.
- BUTLER, RICHARD, Min. Engr., Mines Supt., Lindal Moor Mines,
Ulverston, Lancashire, England.
- CARROLL, DANIEL J. 32 W. 40th St., New York, N. Y.
- CARROLL, FRED, Secy. and Genl. Mgr., Atlas Mining & Milling Co., Sheffels, Colo.
- DAVIES, GEORGE M., Mine Contractor, Lehigh Coal & Navigation Co., Lansford, Pa.
- DEAVITT, WILLIAM JEFFERSON, Min. Engr., Care American Smelting &
Refining Co., Santa Eulalia, Chih., Mexico.
- FOESTER, HALLARD W., Min. Engr., Mill Shift Boss, Lucky Tiger Mining Co.,
Esqueda, Son., Mexico.
- FORT, MICHEL. Director, School of Engineers, Lima, Peru.
- GORDY, SHEPPARD B., Genl. Mine Foreman, Braden Copper Co., Rancagua, Chile.
- GRAHAM, ERNEST R. Supt., Mogul Mining Co., Terry, S. D.
- GUGGENHEIM, SIMON, Chairman, Board of Directors, American Smelt. &
Ref. Co., 120 Broadway, New York, N. Y.
- HENLEY, ARCHIBALD R., Mech. Engr., Lehigh Coal & Navigation Co., Lansford, Pa.
- HILL, STACY H., Mgr., Ingersoll-Rand Co., 717 Providence Bldg., Duluth, Minn.
- HOFFMAN, JOHN F., Min. Engr. 324 Kensington Ave., Salt Lake City, Utah.
- HOLZWORTH, CHARLES R., Asst. Supt., Eliza Blast Furnaces,
Jones & Laughlin Steel Co., Pittsburgh, Pa.
- KAEDING, C. D., Vice-Pres. and Genl. Mgr., Dome Mines Co., Ltd.,
So. Porcupine, Ont., Canada.
- LANE, JAMES A. Mineral Products Co., Marysvale, Utah.
- LEDNUM, EDMUND TOWNSEND, E. I. du Pont de Nemours Powder Co.,
1807 McCormick Bldg., Chicago, Ill.
- LENNOX, LORING COWGILL, Genl. Mgr., Strong Mining Co., Colorado Springs, Colo.
- MARTIN, JOHN DOUGLAS, Asst. Supt., Zinc Smelter, Granby Min. & Smelt. Co.,
East St. Louis, Ill.
- MARTIN, OLIVER CURTIS, Met., Head of Anode and Wire Bar Depts.,
Nichols Copper Co., Laurel Hill, N. Y.
- OLIVER, EARL. Inspector of Oil Properties, Bartlesville, Okla.
- RAMSEY, ELMER RANDALL, Engr., Dorr Cyanide Machinery Co.,
730 First National Bank Bldg., Denver, Colo.
- REED, CHARLES JOHN, Chem. and Met. Engr., 507 Brannan St., San Francisco, Cal.
- ROBERTS, THOMAS S., Underground Supt., Chateaugay Ore & Iron Co.,
Lyon Mountain, N. Y.
- SAMPSON, CARLOS EUGENE. Supt., Recovery Oil Co., Shale, Cal.
- SYKES, WILFRED, Elec. Engr. Westinghouse Electric & Mfg. Co., Pittsburgh, Pa.
- THOENEN, JOHN ROY, Mine Supt., Mond Nickel Co., Ltd., Garson, Ont., Canada.
- TOENGES, ALBERT LOUIS, Mine Foreman; Min. Engr., Box 598, Flat River, Mo.
- VANSANT, WILLIAM FREDERICK, Prest. and Genl. Mgr., Copeland Ore
Sampling Co., Victor, Colo.

Associate Members

- BONSACK, ARTHUR A. Mgr., Ingersoll-Rand Co., 200 N. Broadway, St. Louis, Mo.
- BORNHOFT, HENRY. Asst. Genl. Mgr., Penoles Min. Co., El Paso, Tex.
- COUSINS, GEORGE THOMAS, Mech. Engr., Care Ingersoll-Rand Co.,
11 Broadway, New York, N. Y.
- SCANTLIN, WILLIAM LINGLE. Sales Engr., Ingersoll-Rand Co., Lead, S. D.

Junior Members

BEDWORTH, HERBERT ARTHUR, Student..... Selby, Cal.
 CRISPELL, CHARLES WINEGAR, Student..... 352 Temple St., New Haven, Conn.
 HALL, WILLIAM TEASDALE, Sampling, Care Braden Copper Co., Rancagua, Chile.
 HIGGINS, LANATT TINSLEY..... Care Braden Copper Co., Rancagua, Chile.
 HOFFMAN, LLOYD..... Old Company Club, Lansford, Pa.
 STITT, JAMES BROWNLEE, Min. Engr., Care Braden Copper Co.,
 Sewell, via Rancagua, Chile.
 WIEGAND, AUGUST JOHN, Student..... 56 N. Washington St., Wilkes-Barre, Pa.

Change of Status—Junior Member to Member

PRATT, ALLYNE FRANCIS..... 727 E. 43d St. N., Portland, Ore.

CANDIDATES FOR MEMBERSHIP

APPLICATION FOR MEMBERSHIP.—The Institute desires to extend its privileges to every person to whom it can be of service. On the other hand, it is not desirable that persons should be admitted to membership in classes for which they are not qualified. Members of the Institute can be of great service if they will make a practice of glancing through the list of applicants and promptly notifying the Committee on Membership, or the Secretary of the Institute, of any persons whom they think should not be classified in accordance with the list given.

The following persons have been proposed during the period Aug. 10 to Sept. 10, 1915, for election as members of the Institute. Their names are published for the information of Members and Associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Board of Directors, which has the power of final election.

Members

Emerson Warren Armstrong, La Libertad, Nicaragua.

Proposed by Frank H. Probert, W. G. McBride, S. M. Parker.

Born 1883, Aughtim, Ont., Canada. 1900, Educated, London, Ont., Canada. 1901, Assayer, Silver City Assay Office, Silver City, N. M. 1901, Shift Boss, Concentrator, Faywood Lead Co., Cooks Peak, N. M. 1902, Chem., Copper Queen Cons. Min. Co., Bisbee, Ariz. 1902, Chem., Black Diamond M. & S. Co., Black Diamond, Ariz. 1903, Assayer, G. C. Clark, Bisbee, Ariz. 1904–05, Engr., Clark Engrg. Co., Bisbee, Ariz. 1906, Engr., Franklin W. Smith & Co., Bisbee, Ariz. 1907, Engr., Hovland & Smith, Globe, Ariz. 1908, Engr., Black Warrior Copper Co., Globe, Ariz. 1909–10, Assaying and Engrg., Bragonier & Armstrong, Parker, Ariz. 1911, Millman, Minas del Tajo, Rosario, Sinaloa, Mexico. 1912, Assayer, Albion-Millett Min. Co., Millett, Nev. 1912, Millman, Associated Milling Co., Manhattan, Nev. 1913, Ore Shipping, Codd Lease, Rochester, Nev. 1914, County Surveyor's office, Bisbee, Ariz.

Present position: Met., Babilonia Mines, Ltd.

Frank Basil Anderson, Bisbee, Ariz.

Proposed by Arthur Notman, Gerald Sherman, Alexander V. Dye.

Born 1885, Stockholm, Sweden. 1890–99, Grammar school; 1899–1900, High school, Rapid City, S. D. 1900–03, Preparatory school, S. D. School of Mines. 1903–07, S. D. School of Mines; B. S. 1907, Assayer, Ely Bonanza Min. Co., Ely, Nev.; Surveyor, U. S. Reclamation Service, Yuma, Ariz. 1908–09, Assayer and Cyanider, Lucky Strike Min. Co., Novak, S. D. 1909–10, Assayer, Climax Min. Co., Bourne, Ore. 1910–12, Assayer and Chem., Wells & Co., Portland, Ore.

Present position: 1912 to date, Sampler, Copper Queen Cons. Min. Co.

Lewis Newton Bailey, Kellogg, Ida.

Proposed by J. W. Gwinn, Stanley A. Easton, James S. Wyatt.

Born 1884, San Diego, Cal. 1906, Univ. of California; B. S. 1907, North Star Mines Co., Grass Valley, Cal. 1908-09, Montana-Tonopah Co., Tonopah, Nev. Present position: 1910 to date, Mill Supt., Ontario Min. Co.; Asst. Mill Supt., Bunker Hill & Sullivan Min. Co.

Ernest Barth, Tulsa, Okla.

Proposed by M. M. Valerius, F. Julius Fohs, G. H. Cox.

Born 1887, Aadorf, Switzerland. 1908, Grad., Technical School, Burgdorf, Switzerland. 1913, Grad., Univ. of Berne, Switzerland, in Geology and Chemistry; Ph. D. 1913, Appointed as a member of the geological staff, Union des Pétroles d'Oklahoma, Paris, France, to perform the practical field work with the Oklahoma State Oil Co., Tulsa, which is the American end of the French company.

Present position: Consulting Oil Geologist.

Daniel Deronda Berolzheimer, New York, N. Y.

Proposed by Maximilian Toch, Ray P. Perry, E. Gymbon Spilsbury.

Born 1877, New York, N. Y. 1887-93, Public schools; 1893-95, College City of New York; 1895-1900, New York University, New York, N. Y. 1897-1900, Univ. of Wuerzburg, Wuerzburg, Bavaria, Germany. 1900-04, Chemist in charge, biochemical and analytical laboratory of Dr. Judson Daland, Philadelphia, Pa. 1904-06, Private research and analytical work. 1906-10, Chemist in charge, analytical laboratory, Arlington Chemical Co., Yonkers, N. Y. 1910-11, Librarian, American Chemical Society, New York. 1910-15, Librarian, The Chemists' Club, New York. Present position: Librarian, Barrett Manufacturing Co.

Edward Anderson Blanton, Jr., Philadelphia, Pa.

Proposed by W. L. Saunders, L. D. Albin, George A. Howells.

Born 1858, Locust Grove, Va. 1865-70, Log Cabin Schools, Amelia Co., Va. 1870-72, Private school, Philadelphia, Pa. 1872-83, Practical mechanics, Farmville, Va. 1883-88, Charge of Howard Foundry & Machinery Works, Philadelphia, Pa. 1888-91, Chief Mech. Engr., La Union and El Callao Gold Min. Co., Venezuela. 1893, Mfg. general line of machinery, Howard Foundry & Machine Works, Philadelphia, Pa. 1893-96, Chief Engr., Crown Reef Gold Min. Co., Johannesburg, S. Africa, and Asst. Resident Mgr., Buffetsdoorn Estate & Gold Min. Co., S. Africa. 1897, Invented Blanton self-tightening cam and cam shaft. 1910, Franklin Institute, Philadelphia, Pa., Certificate of Merit. 1896-99, Chief Engr., New Modderfontein Gold Min. Co., S. Africa. 1899-1901, Machinery for Min., Bourse, Philadelphia. Present position: 1901 to date, Pres., Blanton Copper Min. Syndicate.

Ygnacio Safford Bonillas, Bisbee, Ariz.

Proposed by Arthur Notman, Alexander V. Dye, Gerald Sherman.

Born 1887, Tucson, Ariz. 1908, Mass. Inst. of Tech.; B. S. 1908-12, Asst. Geol. and Geo., Instituto Geologico Nacional, Mexico, D. F., Mexico. 1912, Independent mining practice in Mexico.

Present position: 1913 to date, Geol., Copper Queen Cons. Min. Co.

George Oliver Bradley, San Francisco, Cal.

Proposed by Charles W. Merrill, M. H. Kuryla, Abbot A. Hanks.

Born 1867, Colorado. 1887, Maid of Erin Min. Co., Leadville, Colo. 1889, Elkhorn Mine, Mont. 1890, Anaconda Copper Co., Mont. 1893, Copper Queen Cons. Copper Co., Bisbee, Ariz. 1895, United Verde Copper Co., Jerome, Ariz. 1896, Detroit Copper Co., Morenci, Ariz. 1898, De Lamors Golden Gate, Mercer, Utah. 1904, Utah Copper Co., Utah. 1908, Pittsburgh Silver Peak G. M. Co., Nevada. 1909, Ray Cons., Chino Copper. 1915, Butte Superior Copper Co., Alaska Gastineau Min. Co. and others.

Present position: Design and construction of plant, Alaska Juneau G. M. Co.

Charles Arnold Cabell, Carbon, W. Va.

Proposed by J. S. Cunningham, C. E. Krebs, Carl Scholz.

Born 1870, Charleston, W. Va. Free School of Charleston, W. Va. Three years in the Kanawha Military Inst. of Charleston, W. Va. 1884-90, Mt. Carbon Co., Ltd., Powellton, W. Va. 1890-1902, Carbon Coal & Coke Co., Carbon, W. Va. Present position: 1902 to date, Vice-Prest., Carbon Coal & Coke Co.

Lewis Carl Chapman, Tecumseh, Neb.

Proposed by E. G. Woodruff, David T. Day, Van H. Manning.

Born 1890, Tecumseh, Neb. 1904-08, Tecumseh High School. 1910-14, Dept. of Geology, Univ. of Neb. 1909, U. S. Forest Service, Dist. 2, Wyo. and Colo. 1910-11, summers, U. S. Geological Survey, in N. M. and Wyo. 1912-13, Geology work, Pearson Co., in Mexican oil fields. 1914, Geol. (Gypsy Oil Co.), Gulf Co., in Okla.

Present position: Geol., Producers Oil Co., Houston, Tex., Shreveport, La.

LaVern John Charles, Elephant Butte, N. M.

Proposed by Harry J. Wolf, W. Ray Cox, R. B. Paul, Arthur J. Hiester, E. W. Keith.

Born 1877, Morrison, Colo. Common and high school education. 1902, Colorado School of Mines; E. M. 1902, Editor, Mining Reporter, Denver, Colo. 1902-15, Asst. Engr., U. S. R. S. 1908, Assoc. Memb.; 1914, Memb. A. S. C. E.

Present position: Construction Engr., Elephant Butte Dam.

Ervin Henry Clausen, Arroyo, Masbate, P. I.

Proposed by Ernest A. Hersam, W. S. Morley, Andrew C. Lawson.

Born 1889, Napa, Cal. To 1907, Public schools, San Francisco. 1908, Heald's Business College, San Francisco. 1909-13, Univ. of California; B. S., College of Mining. 1908, Salesman, G. M. Hickman Hardware Co., San Francisco. 1909, Bank clerk, Mission Bank, San Francisco. 1912, Machinist's helper, Union Iron Works, San Francisco. 1913-14, Assayer and Chemist, Penn Mining Co., Campo Seco, Cal.

Present position: 1915 to date, Min. Engr., Colorado Mining Co.

Simeon K. Dahl, Messina, Transvaal, So. Africa.

Proposed by Milo W. Krejci, Willis T. Burns, A. B. Emery.

Born 1881, Rodx, Norway. 1896, Finished high school. 1903, Preparatory college. 1906, Grad., Technical College at Christiania, Norway; M. E. 1906, Draftsman, Illinois Steel Works. 1906-07, Designer, Sargent & Lundy, Chicago. 1907-08, Designer and checker, B. & M. Reduction Works, Great Falls, Mont. 1908-10, Designer and checker, Int. Smelt. & Ref. Co.; building of Tooele Plant. 1910-11, Mech. Engr., A. S. & R. Co., Velardena, Mexico. 1911-12, Supt. of Construction and Master Mechanic, Kennecott Mines Co., Kennecott, Alaska. 1912-13, Engr. in charge, La Fé, Guadalajara, Zac., Mexico. 1913-14, Handy Man, Federal Lead Co., Flat River, Mo.

Present position: Mill Supt., Messina Transvaal Dev. Co.

Clarence Benjamin Dutton, Washington, D. C.

Proposed by Van H. Manning, W. A. Williams, David T. Day.

Born 1885, Boston, Mass. 1902, Grad., high school, Medford, Mass. 1903, Post-graduate work, preparatory for Harvard Univ.; entered with class of 1904, but left before end first semester on account of death of father. 1905-06, Studied law, Boston Evening Law School. 1907-11, Studied law, George Washington Univ., Washington, D. C.; LL.B. 1912, Special courses, Geology and Mineralogy, George Washington Univ. 1913 to date, Special courses of reading economic geology, metallurgy, etc. 1907-11, Law Clerk, Bureau of Immigration, Washington, D. C. 1911-15, Law Clerk and Law Examiner, Bureau of Mines, Washington, D. C.; engaged on investigation work, oil fields, Okla.; copper and lead smelters, west of Mississippi; assisted in compilation U. S. Mining Laws and their annotation; have worked on legislative matters affecting mineral industries, such as Alaska leasing act (coal), etc.; assisted W. R. Ingalls and other members of Committee of Amer. Inst. of Min. Engrs. in preparation model Metal Mines Mining Law. Made special study foreign mining laws, etc.

Present position: Law Examiner, Bureau of Mines.

Frederick Herman Gartung, Miami, Okla.

Proposed by C. A. Wright, H. A. Buehler, O. M. Bilharz.

Born 1888, Bay City, Mich. 1906, Grad., Bay City Eastern High School. 1911, Grad., Michigan College of Mines; E. M. 1911, R. S. Fitch Engrg. Co., Joplin, Mo. 1911, F. H. Gartung, Min. Engr., Successor to R. S. Fitch Engrg. Co. 1913, Engr., T. F. Lennan.

Present position: 1914 to date, Supt., Lennan Zinc & Lead Co., Miami Zinc & Lead Co., and Quapau Min. Co.

Alexander R. Globe, Timmins, Ont., Canada.

Proposed by H. W. Hardinge, P. A. Robbins, Charles Humphrey.

Born 1880, Westfield, N. B., Canada. General public school education. 1902-06, Surveyor, DeBeers Cons. Mines, Ltd., Kimberley, So. Africa. 1906-08, Asst. Chief Min. Engr., DeBeers Cons. Mines, Ltd. 1908-11, Mill Supt., McKinley Darragh Savage Mines, Cobalt, Ont., Canada. 1911-14, Supt., Hollinger Gold Mines, Ltd., Timmins, Ont., Canada.

Present position: 1914 to date, Asst. Genl. Mgr., Canadian Min. & Furnace Co.

W. B. Gohring, Warren, Ariz.

Proposed by Arthur Notman, Gerald Sherman, Alexander V. Dye.

Born 1877, Boston, Mass. 1903, Harvard; S. B. 1904-08, Geol., Calumet & Arizona Min. Co.

Present position: 1908 to date, Supt. of Mines, Calumet & Arizona Min. Co.

Clarence William Lawr, El Tigre, Son., Mexico.

Proposed by L. R. Budrow, R. T. Mishler, James W. Malcolmson.

Born 1879, Traverse City, Mich. 1899, Grad., High School, Elk Rapids, Mich. 1899, Entered Law Dept., Univ. of Michigan, Ann Arbor, Mich. 1900-01, Remained out of college on account of lack of funds. 1901-02, Re-entered Univ., matriculating in the Dept. of Literature, Science and Arts. 1905, Grad., Univ. of Michigan; A. B., after having specialized the four years in chemistry. 1906-07, Asst. Chem., United Zinc & Chemical Co., Argentine, Kan. 1907-08, Chief Chem., Pan-handle Smelt. & Ref. Co., Ponderay, Ida.; smelting company went out of business. 1908-09, Head Chem., Silver Fissure Min. Co., Polaris, Mont. 1910-11, Head Chem., Smith, Emory & Co., Los Angeles, Cal.

Present position: 1913 to date, Chem., Tigre Min. Co.

Raymond George Lawry, Chicago, Ill.

Proposed by Newell G. Alford, Howard N. Eavenson, E. O'Toole.

Born 1879, Galesburg, Ill. 1903, Kansas State Agricultural College; B. S. in Mech. Engrg. 1901-03, with Frank D. Moore, Consulting Engr. 1903, short period, Webster Mfg. Co. and American Steel & Wire Co. 1903-15, Roberts & Schaefer Co.; in charge of designing of coal mine plant structure.

Present position: Designing and contracting engineer.

James Bradford Lewis, Jr., Warren, Ariz.

Proposed by Arthur Notman, Gerald Sherman, Alexander V. Dye.

Born 1882, Walpole, Mass. 1906, Harvard; S. B. 1906-07, Pike Hill Mines, Corinth, Vt. 1908, Tenn. Copper, Iron & Sulphur Co., Ducktown, Tenn.

Present position: 1908 to date, Engr. Dept., Calumet & Arizona Min. Co.

Robert Earl McConnell, Los Angeles, Cal.

Proposed by Seeley W. Mudd, Frank A. Keith, C. O. Lindberg.

Born 1889, Montrose, Colo. 1894-1905, Public schools, Durango, Colo. 1905-07, Univ. of Colo. 1907-10, Columbia Univ.; E. M. 1910-11, Asst. Engr., Chino Copper Co., Santa Rita, N. M.

Present position: 1912 to date, Min. Engr., for S. W. Mudd.

James Comley McCoy, Nayatt, Bristol Co., R. I.

Proposed by L. V. Emanuel, F. L. Antisell, S. Skowronski.

Born 1862, Urbana, Ill. Common and high school education with one year at Wesleyan Univ., Bloomington, Ill. 1896, Mgr., New England Electrolytic Copper Min. Co., Central Falls, R. I. 1898-1905, Mgr., Raritan Copper Works, Perth Amboy, N. J.

Present position: 1905 to date, Mgr., mining properties and mills.

Warner McLaughlin, Cobalt, Ont., Canada.

Proposed by James Johnston, Fred W. Young, H. A. Kee.

Born 1888, Port Henry, N. Y. 1898-1903, Collegiate School, New York. 1903-06, St. Paul's School, Concord, N. H. 1906-09, Harvard Univ.; A. B. 1909-13, Columbia Univ.; Met. E. 1913-14, Smelterman, Sampler, Canadian Copper Co., Copper Cliff, Ont.

Present position: 1914 to date, Asst. Met. Engr., Nipissing Min. Co.

DeWitt Clinton MacKallor, Joplin, Mo.

Proposed by J. F. Kemp, Charles P. Berkey, Alfred J. Moses

Born 1884, Short Hills, N. J. 1890-98, Public schools, Milburn, N. J. 1899-1901, High school, Short Hills, N. J. 1902, DeWitt Clinton High School, New York. 1903-06, Columbia Univ.; E. M. 1906, Trammings, machine drilling, surveying, Tennessee Copper Co., Ducktown, Tenn. 1906-09, Cyanide shift boss, timekeeper, mine shift boss, Engr., Dolores Mines Co., Dolores, Chih., Mexico. 1909-10, Mgr., Duria de Oro Gold Min. Co., Duria de Oro, Chih., Mexico. 1910-11, Engr., Calabacillas Min. Co., Calabacillas, Chih., Mexico. 1911-12, Mine Foreman, Dolores Mines Co. 1912-13, Asst. Mgr., Duria de Oro Gold Min. Co. 1913-14, Engr., Chisos Min. Co., Terlingua, Tex. 1914-15, Mgr., Cossak Min. Co., Bland, N. M.

Present position: With H. R. Conklin on experimental work.

James Donovan Malcolmson, Douglas, Ariz.

Proposed by L. R. Budrow, R. T. Mishler, Edward L. Dufourcq.

Born 1890, Anganguero, Mexico. 1913, Engrg. School, Univ. of Kansas; B. S. 1913, Purchasing Agent, Choctaw Portland Cement Co., Hartshorne, Okla. 1914, Secy., Rosedale Crushed Stone Co., Kansas City, Mo.

Present position: Purchasing Agent and Costkeeper, Tigre Mining Co., Esqueda, Sonora, Mexico.

William R. Maycumber, Esqueda, Sonora, Mexico.

Proposed by L. R. Budrow, R. T. Mishler, Edward L. Dufourcq.

Born 1881, Weedsport, N. Y. 1898, Grad., Weedsport High School, Weedsport, N. Y. Hold academic diploma of the Regents of the University of the State of N. Y. Mech. Course, International Correspondence Schools. 1898-1900, Apprentice machinist, Stearns Bicycle Co., Syracuse, N. Y. 1900-02, in Philadelphia and Standard mills, U. S. Reduction & Refining Co., Colorado City, Colo. 1902-03, Foreman in mill, Penn. Wyoming Copper Co., Grand Encampment, Wyo. 1903-07, Head repair man, shift boss, foreman in mill, Cananea Cons. Copper Co. 1908, Shift boss, Tigre Min. Co. 1908-09, Mill Foreman, Cananea Copper Co.

Present position: 1909 to date, Mill Supt., Tigre Min. Co.

Warren J. Mead, Madison, Wis.

Proposed by Charles F. Rand, Bradley Stoughton, C. K. Leith.

Born 1883, Plymouth, Wis. 1906, Univ. of Wisconsin; B. S. 1908, M. A. 1906-08, U. S. Geological Survey, Lake Superior division. 1908-15, Consulting Geological work in Lake Superior iron and copper districts, Laton iron districts, Arkansas bauxite fields, Butte, and other localities.

Present position: 1906 to date, Asst. Prof. of Geology, Univ. of Wisconsin.

Charles Whiting Newton, Wallace, Ida.

Proposed by Walter Harvey Weed, Theodore Simons, Paul A. Gow.

Born 1870, Superior, Wis. 1888, Finished General High School, Superior, Wis. 1889, Technical work under tutor, Superior, Wis. 1890, Draftsman, W. H. Newton Land Co. 1891, Running transit, Beck & Smith Engrg. Co. 1892-93, Drafting in architect's office. 1894, Supt., Doly Min. Co., Ely, Minn. 1907-15, Supt., Butte Ballaklava Copper Co., Butte, Mont., and Butte Zenith City Min. Co.

Present position: Supt., Cons. Interstate Callahan Min. Co.

Iichiro Omori, Kumamoto, Japan.

Proposed by Benzo Katsura, Tadashi Inouye, Takichi Ohashi.

Born 1879, Kumamoto, Japan. 1901, Grad., Kumamoto High Middle School. 1904, Grad., Min. and Met. Dept., Tokyo Imperial Univ., "Kogakushi." 1904, Min. Engr., Mining Bureau of Prince Shimazu.

Present position: 1906 to date, Prof. of Metallurgy, Kumamoto High Technical School.

C. I. Schultz, Nacozari, Son., Mexico.

Proposed by Walter Douglas, J. S. Williams, H. T. Hamilton.

Born 1867, Germany. General education. 1892-1906, Shift Foreman and Genl. Foreman, Boston Montana Copper Co., Butte, Mont. 1906-08, Supt., Butte & Arizona, Hereford, Ariz.

Present position: 1908 to date, Genl. Supt., Moctezuma Copper Co.

Frank Henry Skeels, Cornucopia, Ore.

Proposed by R. G. Amidon, Bancroft Gore, C. H. Bowman.

Born 1884, Uniontown, Wash. 1911, Washington State College; B. S. in Min. Engrg. 1911-12, Chief Engr., Stewart Min. Co., Kellogg, Ida. 1912-13, Supt., Montana Metal Min. Co., Helena, Mont. 1913-14, Const. Engr. and Mine Supt., August Min. Co., Landusky, Mont. 1914-15, Cons. Engr., W. E. Cullen, Spokane, Wash.

Present position: Mgr., Red Mountain Min. Co.

James R. Stack, Perth Amboy, N. J.

Proposed by Clarence P. Linville, Harry H. Alexander, F. C. Newton.

Born 1889, Vermont. 1912, Univ. of Vermont; B. S. 1912, Grasselli Chemical Co.

Present position: 1913 to date, Chem., Amer. Smelt. & Ref. Co.

Melville Boicourt Stewart, Pekin, Ill.

Proposed by H. H. Stoek, F. W. DeWolf, S. O. Andros.

Born 1888, Metropolis, Ill. 1902-06, Metropolis High School. 1906-11, Univ. of Ill. Combination Civil and Mining Engineering.

Present position: 1911 to date, Min. Engr., Tazewell Coal Co., Pekin, Ill.; Roanoke Coal Co., Roanoke, Ill.; Johnston City Coal Co., Johnston City, Ill.

Byron A. Stimmel, Trail, B. C., Canada.

Proposed by R. H. Stewart, S. S. Fowler, Francis A. Thomson.

Born 1877, Salina, Kan. 1889, Left grammar school, Salina, Kan. 1890-96, Grammar and high schools, Spokane, Wash. 1896-1905, Grad., Washington State College, Pullman, Wash., Mining Dept., with work in various mines and smelters. 1905, Helper in electrical shops, Granby Min. & Smelt. Co., Grand Forks, B. C., Canada. 1906-07, Assayer, Dominion Copper Co., Boundary Falls, B. C.

Present position: 1908 to date, Asst. Met., Consolidated Min. & Smelt. Co.

Frederick Thomas Teddy, Ducktown, Tenn.

Proposed by M. A. Caine, T. W. Caver, J. B. Risque.

Born 1884, Redruth, England. 1889-93, English private and public schools. 1893-1901, Grammar and high school, Ishpeming, Mich. 1901-03, Engrg. Course, International Correspondence School. 1906-08, Mathematics, American School of Correspondence. 1908-12, Michigan College of Mines; B. S. and E. M. 1901-03, Apprentice mechanic, W. J. Roberts. 1903-05, Mechanic, Tri-Bullion Smelt. & Dev. Co. 1905-07, Mechanic, Cleveland-Cliffs Iron Co. 1911-12, summer, Asst. to Prof. of Mech. Engrg., Mich. College of Mines. 1912, Engr., Railway Valuation, Charles Hansel & Co. 1912-14, Min. Engr., Tenn. Copper Co., Ducktown. 1914-15, High School Instructor, science and manual arts. 1915, Engr., Chattanooga Copper Co.

Present position: Min. Engr., Tenn. Copper Co.

Karl Saben Twitchell, Nicosia, Island of Cyprus.

Proposed by Alvin B. Carpenter, Seeley W. Mudd, C. O. Lindberg.

Born 1885, St. Albans, Vt. 1904, St. Albans High School, Vt. 1908, Kingston School of Mining, Green's Univ., Kingston, Ont.; B. Sc. 1909, Had done required amount of practical work and obtained degree M. E. 1908-10, Engr., De Lamar Co., Ltd., De Lamar, Idaho. 1910-12, Supt., Red Ledge Mine, De Lamar Co., Ltd., Washington, Cal. 1912-13, Engr., De Lamar Co., De Lamar, Idaho. 1913-14, Asst. Supt., Flint Mines, Ltd., Flint, Idaho, for Bainbridge Seymour Co., London. 1914, Sampler, Mary Murphy mine, Romley, Colo. 1915, spring, Asst. on examinations in New Mexico and Arizona.

Present position: Mine Foreman.

Charles Howard Urquhart, Joplin, Mo.

Proposed by C. A. Wright, H. A. Buehler, O. M. Bilharz.

Born 1874, Leominster, Mass. General school education. 1897-1907, Mining in British Columbia, Alaska, Nevada, and Mexico. 1908-09, Cyanide Supt. (Concentrate plant), Black Oak Mine, Cal. 1910-12, Designer and Supt. of mill (new) Black Oak Dev. Co., Cal. 1913-14, Met. testing, etc.; incidentally at Golden Cycle Mill, Colo. 1914-15, Met. Engrg. and mill building, Joplin, Mo.

Present position: Metallurgical Engrg. Unattached.

William A. Wasley, Esqueda, Sonora, Mexico.

Proposed by L. R. Budrow, R. T. Mishler, Charles M. Heron.

Born 1886, Platteville, Colo. 1905, Grad., Greeley High School, Greeley, Colo. 1909, Grad., Colorado School of Mines; E. M. 1909-10, in mill, Utah Copper Co., Garfield, Utah. 1910-11, Chemist, Garfield Smelt. Co. 1911-12, Miami Copper Co., Miami, Ariz. Ore representative, Cananea Copper Co., Cananea, Sonora, Mexico. Also in mill at Miami. 1912-13, in concentrator, Cananea Cons. Copper Co., Cananea, Sonora, Mexico.

Present position: Cyanide Foreman, El Tigre Mining Co.

Junior Member to Member

William A. Wong, Tayeh, Hupeh, China.

Became a Junior Member in 1913. Out of college and actively engaged in mining for more than a year.

Present position: Engr. in charge of an iron mine.

Change of Status—Associate to Member

Albert H. Anderson, Worcester, Mass.

Born 1889, Worcester, Mass.

Since election as Associate in 1913, have been in engineering and operating control of an abrasive milling plant, employing 100 men.

Present position: Abrasive Mill Foreman.

CHANGES OF ADDRESS OF MEMBERS

The following changes of address of members have been received at the Secretary's office during the period Aug. 10, to Sept. 10, 1915. This list, together with the list published in *Bulletin* Nos. 100 to 105, April to September, 1915, and the foregoing list of new members, therefore, supplements the annual list of members corrected to Mar. 1, 1915, and brings it up to the date of Sept. 10, 1915.

ADAMS, HUNTINGTON.....	Care White, Weld & Co., 14 Wall St., New York, N. Y.
ALEXANDER, WILLIAM H.....	74 Broadway, New York, N. Y.
ALLEN, ROY H.....	Lunenburg, Mass.
ARNOLD, JOHN B.....	Suite 840, 200 Fifth Ave., New York, N. Y.
BAGGE, N. O.....	46 Cedar St., New York, N. Y.
BAILLIE, FRANK S.....	Rand Building, Baker, Ore.
BIBLE, JOSEPH W.....	Hanover, N. M.
BLACKMAR, FRANK H.....	2924 Garfield Ave., Kansas City, Mo.
BLAYLOCK, D. W.....	Flat River, Mo.
BORDEAUX, ALBERT F. J.....	29 Rue Faucigny, Fribourg, Switzerland.
BRINSMADE, ROBERT B.....	9a Galeana 1, Puebla, Pue., Mexico, via Vera Cruz.
BURGER, C. C.....	Room 2050, 42 Broadway, New York, N. Y.
BURLINGAME, WALTER E.....	1100 Lafayette St., Denver, Colo.
CLARK, DOUGLAS.....	Stanford University, Cal.
CLEMENT, HARRISON E., Engr. and Contractor,	Care Milton D. Grosh,
	University Club, Salt Lake City, Utah.
COCHRAN, RALF S., Met.....	Republic Iron & Steel Co., Youngstown, Ohio.
COOKE, JOSEPH H., Mono Development Co.,	625 Dooly Bldg., Salt Lake City, Utah.
CRABTREE, FRED.....	Carnegie Institute of Technology, Pittsburgh, Pa.
CUNNINGHAM, P. H.....	1302 Highland Ave., New Castle, Pa.
CUTTING, GEORGE WALFORD.....	Morenci, Ariz.
DAKE, WALTER M., Jr.....	Care Stall Bros., Masonic, Mono Co., Cal.
DOUGAN, LEE DEWANE.....	Elko Prince Leasing Co., Midas, Nev.
DUNKIN, DAMON D.....	Buck, via Alderson, Okla.
DURHAM, EDWARD B.....	2227 Ward St., Berkeley, Cal.
EDELEN, A. W., Amer. Smelt. & Ref. Co.,	200 Mutual Bldg., Mexico City, Mexico.
ELDREDGE, ROBERT B.....	911 W. Pike's Peak Ave., Colorado Springs, Colo.
EMANUEL, L. V.....	Care River Smelt. & Refining Co., Florence, Colo.
ESSELSTYN, J. N.....	Golden Pond, Ky.
EYE, C. M., Supt.....	Benguet Cons. Mining Co., Baguio, Benguet Prov., P. I.
FAHRENWALD, F. A., Care Mining Dept.,	Case School of Applied Science,
	Cleveland, Ohio.

- FAIRBAIRN, GEORGE, 18 Welbeck Court, Addison Bridge Place, Kensington,
London, W., England.
- FRIEDRICH, ALFRED KARL.....Care Empire Copper Co., Enaville, Idaho.
- GABY, WALTER E.....314 W. Granite St., Butte, Mont.
- GOODRICH, R. R.....Care University of Idaho, Moscow, Idaho.
- GRANTING, HUGO GEORGE.....Box 423, Golden, Colo.
- GRONLUS, HEINRICH F.....Huerfanos 1326, Santiago de Chile, Chile.
- GUGGENHEIM, E. A.....Elm Point, Great Neck, L. I.
- HALL, MORTIMER L., Min. Engr., Intermountain Copper Mining Co.,
Iron Mountain, Mont.
- HANCE, JAMES H.....General Delivery, Iowa City, Iowa.
- HARDER, EDMUND C., Geol.....U. S. Geological Survey, Washington, D. C.
- HASTINGS, JOHN HARTLEY.....212 Vine St., Monongahela City, Pa.
- HAVLIN, THOMAS NEWELL.....Care Havlin & Ellis, Box 346, Commerce, Okla.
- HEBERLEIN, KUNO B., Care American Metal Co., 61 Broadway, New York, N. Y.
- HEROLD, STANLEY CARROLLTON.....Instructed to hold all mail.
- HOCKING, RICHARD O., Inspector of Mining Operations, Care Meriden Iron Co.,
Hibbing, Minn.
- HUMMELL, A. S.....Care Jones & Laughlin Steel Co., Woodlawn, Pa.
- HLISENG, AXEL O.....Joplin, Mo.
- IMHOFF, ALEXANDER.....Gold Road, Ariz.
- IVBY, JOSEPH H.....Casillas 855-856, Antofagasta, Chile.
- JARVIS, ROYAL P., Chief Chem., Cananea Cons. Copper Co., Cananea, Son., Mexico.
- JOHNSON, FREDERICK ARTHUR.....215 W. 23d St., New York, N. Y.
- JONES, ALEXANDER B., Accountant, Cobriza Mines Development Corp.,
Johnson, Ariz.
- JUDSON, WILBER.....602 N. Washington Ave., Lansing, Mich.
- KASAHARA, WASHITARO, Chief Engr., Hibira Copper Mine, Higashi, Usukigori,
Hyuga, Japan.
- KEENEY, ROBERT M.....Care Standard Chemical Co., Canonsburg, Pa.
- LEWIS, ROBERT S.....University of Utah, Salt Lake City, Utah.
- LEWYN, OSWALD MARK, Union Minière Co., Friars House, New Broad St.,
London, England.
- LINDSAY, WILLIAM RUFUS.....Box 582, Juneau, Alaska.
- McINTOSH, FRANK K.....Brownsville, Ind.
- MANNERS, ERNEST A., 781-82 Salisbury House, London Wall, London, E. C.,
England.
- MASTERS, HARRIS K.....1403 Fairfield Ave., Bridgeport, Conn.
- MAXON, W. L.....Anaconda, Mont.
- MOORE, BLISS.....2111 N. Nevada St., Colorado Springs, Colo.
- MORRIS, RICHARD J.....Church House, 12th and Walnut Sts., Philadelphia, Pa.
- MORROW, JOHN T., Factory Products Export Corp., 50 Broad St., New York, N. Y.
- MUSSIGBROD, LUDWIG S.....Bearmouth, Mont.
- NEWTON, FRANCIS CLEMENT.....Care Newton Manufacturing Co., Lowell, Mass.
- OFFICER, HERBERT GEORGE.....Panulcillo, via Coquimbo, Chile.
- PALLISTER, HUGH D., Prof. of Geology and Mining, State School of Mines,
University of Texas, El Paso, Texas.
- PALMER, WILLIAM J. A.....Remlig, Tex.
- PEARSON, JAMES.....Lehigh, Mont.
- PFORDE, OTTO F.....Cairo, Greene Co., N. Y.
- POCOCK, CECIL W.....Northern Ore Co., Edwards, N. Y.
- PRINCE, ERNEST.....Boatmen's Bank Bldg., St. Louis, Mo.
- PROBERT, FRANK H.....Hobart Bldg., San Francisco, Cal.
- PRYER, W. CRISTY.....522 6th Ave. S. E., Aberdeen, S. D.
- RADCLIFFE, DONALD HEWSON, Instructor in Geology and Mineralogy,
Missouri School of Mines, Rolla, Mo.
- RAINSFORD, RALPH S.....Care General Motors Co., Detroit, Mich.
- RALSTON, WILLIAM CHAPMAN.....315 Bush St., San Francisco, Cal.
- RAWLINGS, STUART L.....704 West Coast Life Bldg., 354 Pine St., San Francisco, Cal.
- RODGERS, M. K.....R. F. D. 1, Box 92, Brentwood Park, Sawtelle, Cal.
- ROGERS, BENJAMIN C.....Care Standard Mine, Box 427, Metcalf, Ariz.
- SAYRE, HERBERT A.....1530 22d St., Des Moines, Iowa.
- SCHMIDT, HENRY C.....170 Calle Bolivar, Monterey, N. L., Mexico.
- SCHRAPPS, PAUL CURTIS.....Care Harry Robinson, Elko, Elko Co., Nev.
- SEDIIV, MILES.....3379 E. 49th St., Cleveland, Ohio.
- SEMPLE, ROBERT ALEXANDER.....380 Lenox Road, Brooklyn, N. Y.
- SEN, JANSI.....Care Wing Wo Chong, 34 Pell St., New York, N. Y.
- SHELDON, GEORGE R.....University Club, Salt Lake City, Utah.

SMALL, HORATIO LEVERETT	Care Staso Milling Co., Poultney, Vt.
SMITH, LYON	Blair, Nev.
SMITH, WALTER F.	Welsbach Co., Gloucester City, N. J.
SOPER, E. K.	Dept. of Mining, University of Idaho, Moscow, Idaho.
SWEETSER, ARTHUR L.	103 Wilson Ave., Rumford, R. I.
THOMAS, JOHN F., Borough Engr.	City Bldg., Farrell, Mercer Co., Pa.
TRISCHKA, CARL	Box 941, Jerome, Ariz.
TUSCHKA, OTTO	Care Hadden Hotel, 111 G. Ave., Douglas, Ariz.
TYSSOWSKI, JOHN	Room 760, 200 Fifth Ave., New York, N. Y.
UMPLEBY, JOSEPH B.	University of California, Berkeley, Cal.
UNDERHILL, JAMES	U. S. Mineral Surveyor, Idaho Springs, Colo.
UTLEY, H. H.	Mgr., River Smelting & Refining Co., Florence, Colo.
WALTERS, MAURICE B.	Standard Mine 5, Silverton, B. C., Canada.
WANG, Y. TSENSHAN	Hartley Hall, Columbia University, New York, N. Y.
WERNECKE, L.	Care Alaska Treadwell Gold Mining Co., Treadwell, Alaska.
WESTON, CHARLES V.	3d Apartment, 3140 Sheridan Road, Chicago, Ill.
WHITAKER, CHARLES N., JR.	Baldwin Park, Cal.
WIGTON, GEORGE HARRISON	Alpine Apartments, Anaconda, Mont.
WINMILL, HALLETT	Care Borneo Co., Ltd., Lakor Lampang, Upper Siam, via Bangkok.
WINSLOW, FRANCIS	Care Alaska Gastineau Mining Co., Thane, Alaska.
WITT, HERBERT NELSON	Bacon Hall, University of California, Berkeley, Cal.
WOOD, FLETCHER H.	Box 176 H, Metcalf, Ariz.
WORTH, J. G., Min. Engr.	Room 1416, 66 Broadway, New York, N. Y.
WREN, ANDREW ADDISON	Ray, Ariz.
WROTH, JAMES S.	500 Copper Ave., Albuquerque, N. M.
ZIMMERMAN, STANLEY HUGHSON	Care Morococha Mining Co., Morococha, Peru.

ADDRESSES OF MEMBERS AND ASSOCIATES WANTED

Name.	Last address of Record, from which Mail has been Returned.
BOISE, CHARLES WATSON	Care Forminière, 8 Montagne du Parc, Brussels, Belgium.
BONSACK, ARTHUR A.	Ingersoll-Rand Co., 200 N. Broadway, St. Louis, Mo.
COOK, PAUL RICHARDSON	4159 Grand Boulevard, Chicago, Ill.
DOWLER, HARRY P.	Penn-Mary Coal Co., Heilwood, Pa.
GEIGER, ARTHUR W.	220 Mills Bldg., San Francisco, Cal.
GRAVES, McDOWELL	949 Atlantic Ave., Long Beach, Cal.
HOLDEN, ROY J.	Blacksburg, Va.
KIRKLAND, T. C., Cia Met. y Refinadora del Pacifico S. A., Fundicion, Son., Mexico.	
LE NOIR, FRANK H.	Douglas, Alaska.
MILLER, FRANK BARTON	Blair, Esmeralda Co., Nev.
MORSE, ROBERT G.	Mass Steel Casting Co., Everett, Mass.
NASH, WILLARD H.	297 Delaware Ave., Buffalo, N. Y.
OLIVER, ROLAND B.	8 Montagne du Parc, Brussels, Belgium.
PRICHARD, WILL A.	Room 711, 111 Broadway, New York, N. Y.
RHODES, W. B.	Golden, Colo.
RUBEL, MILTON L., Pilonas Min. Co., Topia, Dgo., Mex., via Nogales	
SCHINDLER, DONALD F.	215 N. Murray St., Madison, Wis.
SEIBERT, PERCY A.	Hagerstown, Md.
SOMERS, RANSOM E.	804 E. Seneca St., Ithaca, N. Y.
STILLWAGON, SAM C.	329 W. 58th St., New York, N. Y.
SULLIVAN, WILLARD P.	Henry Walks Co., Norfolk, Va.
WELSH, NORVAL J. E.	Organ, P. O., N. M.
WILCOX, RALPH	Curanilahue, Chile.
WRIGHT, JESSE T.	Wallace, Idaho.

NECROLOGY

The deaths of the following members were reported to the Secretary's office during the period Aug. 10 to Sept. 10, 1915:

Date of Election.	Name.	Date of Decease.
1880	*Comstock, Theodore B.	July 26, 1915
1871,	**Hahn, Otto H.	July 26, 1915
1898,	**Hughes, Hugh	

*Member.

**Life Member.

BIOGRAPHICAL NOTICES

Theodore Bryant Comstock, son of Calvin J. and Amelia (Hanford) Comstock, was born July 27, 1849, at Cuyahoga Falls, Ohio, and received his education in the public schools of Ohio and Pennsylvania, and the Pennsylvania State College, where he was graduated as Bachelor of Agriculture in 1868. A supplementary course at Cornell University gave him in 1870 the degree of Bachelor of Science, and the same institution made him Doctor of Science in 1886.

In 1870, at the age of 21, he served as assistant and photographer in the Morgan expedition to Brazil. In 1871 and 1872 he was Professor of Natural Science at Pelham Priory, N. Y.; in 1873 he was geologist of the expedition of Capt. W. A. Jones to northwest Wyoming and the Yellowstone National Park. In 1875 he founded and directed the Kirtland Summer School of Natural History at Cleveland, Ohio, and in the same year he became, and remained for four years, acting Professor of Geology and Paleontology at Cornell University, where he established a department of economic geology. Meanwhile, in 1876, he served as assistant in the Harvard Summer School of Geology; in 1877 he led an expedition to the Northwest Territory of the Dominion of Canada, and in 1878 he published *An Outline of Geology*.

From 1885 to 1889 he was Professor of Mining Engineering and Physics in the University of Illinois; in 1887 and 1888, Assistant State Geologist of Arkansas; from 1889 to 1891, Assistant State Geologist of Texas; in 1891, founder and until 1895 Director of the Arizona School of Mines; from 1893 to 1895, President of the University of Arizona, and in 1894, Director of the Arizona Agricultural Experiment Station. In 1893 he presided as Vice-President over the National Irrigation Congress, held at Los Angeles, Cal.

At various times, Professor Comstock was directing or advising engineer of important mining companies in Colorado, Arizona, and Mexico. From 1901 to 1906 he was a member of the Executive Committee of the California Miners' Association. He was also an original Fellow of the Geological Society of America, a charter member of the Mining and Metallurgical Society of America; in 1901, editor of the *Bulletin of the Southern California Academy of Sciences*. Having established at Los Angeles his headquarters as consulting engineer, he took a public-spirited interest in the progress of that city, and from 1910 to 1912 was Secretary and Chief Engineer of its Board of Public Utilities. At the time of his death he was President of the California Farm and Home Builders.

Professor Comstock joined this Institute in 1880, and maintained throughout his life a lively and loyal interest in its work and welfare. The following is a list of his contributions to the *Transactions*.

Title	Volume	Page	Year
Notes on the Geology and Mineralogy of San Juan County, Colorado.....	xi	165	1882
The Geology and Vein-Structure of Southwestern Colorado.....	xv	218	1886
Mining Engineering at the University of Illinois.....	xv	589	1886
Engineering Relations of the Yellowstone Park.....	xvi	46	1887
Notes on the Region North of the Vermilion Lake District, in British America.....	xvi	109	1887
Hot Spring Formations in Red Mountain District, Colorado..	xvii	261	1888
The Geology and Vein-Phenomena of Arizona.....	xxx	1038	1900
Superficial Blackening and Discoloration of Rocks, Especially in Desert Regions.....	xxxv	1014	1904

He was, in 1912, one of the founders of the Southern California section of the Institute, of which he was the Chairman until forced by failing health to retire from active life. His colleagues have issued in the name of the Institute, and particularly of its Southern California Section, the following sincere and well-deserved tribute to his character and career:

SOUTHERN CALIFORNIA SECTION AMERICAN INSTITUTE OF MINING ENGINEERS

In Memoriam

Whereas, in the passing beyond of our beloved friend and associate, Theodore Bryant Comstock, the American Institute of Mining Engineers, and particularly the Southern California Section and its Executive Committee, of which he was a most active and efficient member until, through ill health, he asked to be excused from such duties, have lost the active service of one whose thought and heart's desire were for the good of his fellowmen and the up-building of our Section,

And Whereas, Our departed brother had greatly endeared himself to his friends and to the community by his widely known ability and his sterling uprightness,

Therefore, Be It Resolved, That the Southern California Section of the American Institute of Mining Engineers does hereby express its deep sense of loss in the death of him who so long was its presiding officer.

And, Be It Further Resolved, That a copy of these resolutions be presented to Mrs. Comstock in token of our deep sympathy and personal regard, and that copies be forwarded to the members of the Southern California Section and to the Officers of the American Institute of Mining Engineers.

LOS ANGELES, CAL.
July 30, 1915.

SEELEY W. MUDD
C. COLCOCK JONES
F. J. H. MERRILL
A. B. CARPENTER
RALPH ARNOLD

Executive Committee

Professor Comstock died July 26, 1915, at his home in Los Angeles, of heart failure following a protracted illness. His wife, to whom he was married in 1880, survives him.

George Ormrod was born July 13, 1839, at Preston, England, where an elder brother of his still resides, and, after serving as Mayor of the town, is chairman of its Select Council, and influentially active in public affairs. When he was less than two years old, his father, George Ormrod, died, and he was taken by his uncle to Manchester, where he lived until he was nineteen, receiving his education at private schools and the Manchester School of Design and Engineering, and working nearly two years in a railway locomotive shop at Gorton. In 1859, he came to the United States, to visit his mother's brother, William Donaldson, then proprietor of a large anthracite colliery at Tamaqua, Pa. Several weeks after his arrival, his uncle was suddenly killed in an accident; he was persuaded by his relatives to remain, and soon became Outside Superintendent of the colliery. His choice of this country as a permanent residence was sealed in 1861 by his marriage to Miss Permilla Johnson of Tamaqua, who, after a wedded life of nearly 51 years, died in 1911, leaving four married daughters and one son, John D. Ormrod, who still survive.

Mr. Ormrod's business career was intimately connected with the development of the anthracite region and the Lehigh Valley—a wonderful history of industrial progress, in which he bore no small part. He was one of those who opened in 1865 and operated for many years the Girard Mammoth colliery at Raven Run, of which he was Manager during

the famous struggle with the Molly Maguires, and afterwards president until in 1879 the colliery was sold to the Philadelphia & Reading Coal & Iron Co. In 1880, with two associates, he leased the Emaus blast furnace, under the firm name of Ormrod, Fisher & Co. In 1883, the firm built the Emaus pipe foundry, where, first under the above name, and, after its incorporation in 1886, under that of the John Donaldson Iron Co., the manufacture of cast-iron pipe and special castings, especially for gas and water mains, has been carried on for more than 30 years. Mr. Ormrod was Manager and Treasurer until 1906, and President and Treasurer after that, until he died, at which time the company owned 71 acres, 36 buildings, its own water supply and electric light and power plant, etc.; employed 500 workmen and produced annually 50,000 tons of castings. The continuous and progressive success of this enterprise is typical of the quiet but wise and efficient man who managed it from the beginning.

Mr. Ormrod was also a leader in another great industry of the Lehigh Valley. In 1893, together with his son-in-law, Thomas D. Whitaker, he organized the Whitaker Cement Co. (now the Alpha Portland Cement Co.), which was the first in New Jersey, and the second in the United States, to manufacture Portland cement by the rotary-kiln process. In 1897, he was one of the founders of the Lehigh Portland Cement Co., now among the largest concerns of this class in the country; and he was an active participant in other important business enterprises. In Allentown, where he resided for the last 34 years of his life, Mr. Ormrod was universally known and loved as a public-spirited and philanthropic citizen. He was a trustee of the Allentown Hospital and of the Young Men's Christian Association; member (and at one time President) of the Livingston Club; director of the Lehigh Valley Trust Co.; member of the Franklin Institute, the Union League Club, and the Manufacturers' Club of Philadelphia, and the Pennsylvania Club of New York; and trustee and senior warden of Grace Episcopal Church, Allentown.

He joined the American Institute of Mining Engineers in 1881, and maintained to the last his interest in its work and welfare, taking part in many "Institute excursions" with his wife, who thus came to share in the cordial esteem with which he was regarded by his fellow-members.

Mr. Ormrod died at his home in Allentown, Pa., June 21, 1915, from a sudden attack of angina pectoris.

EXECUTIVE COMMITTEES OF LOCAL SECTIONS

New York

Meets first Wednesday after first Tuesday of each month.

DAVID H. BROWNE, *Chairman*, JOHN H. JANEWAY, *Vice-Chairman*.
 F. E. PIERCE, *Secretary*, 35 Nassau St., New York, N. Y.
 P. A. MOSMAN, *Treasurer*.

LEWIS W. FRANCIS.

Boston

Meets first Monday of each winter month.

HENRY L. SMYTH, *Chairman*, ALFRED C. LANE, *Vice-Chairman*.
 HENRY A. WENTWORTH, *Secretary-Treasurer*, 60 India St., Boston, Mass.
 ROBERT H. RICHARDS, ALBERT SAUVEUR.

Columbia

Holds four sessions during year. Annual meeting in September or October.

FRANK A. ROSS, *Chairman*, RUSH J. WHITE, *Vice-Chairman*.
 LYNDON K. ARMSTRONG, *Secretary-Treasurer*, P. O. Drawer 2154, Spokane, Wash.
 FREDERIC KEFFER, FRANCIS A. THOMSON.

Puget Sound

Meets second Saturday of each month.

I. F. LAUCKS, *Chairman*, J. F. MENZIES, *Vice-Chairman*.
 GLENVILLE A. COLLINS, *Secretary-Treasurer*, Box 144, Seattle, Wash.
 H. L. MANLEY.

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 RALPH ARNOLD, A. B. CARPENTER.

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 C. LORIMER COLBURN, *Secretary-Treasurer*, 614 Ideal Bldg., Denver, Colo.
 FRED H. BOSTWICK, W. G. SWART.

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FRANK M. SMITH, *Chairman*, JAMES L. BRUCE, *Vice-Chairman*.
 DARSIE C. BARD, *Secretary*, Montana State School of Mines, Butte, Mont.
 FREDERICK LAIST, W. C. SIDERFIN.

San Francisco

Meets second Tuesday of each month.

G. HOWELL CLEVINGER, *Chairman*, C. W. MERRILL, *Vice-Chairman*.
 JAMES C. RAY, *Secretary-Treasurer*, 1235 Webster St., Palo Alto, Cal.
 F. W. BRADLEY, ANDREW C. LAWSON.

Pennsylvania Anthracite Section

R. V. NORRIS, *Chairman*.
 CHARLES F. HUBER, *Vice-Chairman*, W. J. RICHARDS, *Vice-Chairman*,
 EDWIN LUDLOW, *Vice-Chairman*, ARTHUR H. STORRS, *Vice-Chairman*.
 CHARLES ENZIAN, *Secretary-Treasurer*, U. S. Bureau of Mines, Wilkes-Barre, Pa.
 DOUGLAS BUNTING, FRANK A. HILL, ALBERT B. JESSUP.
 RUFUS J. POSTER, JOHN M. HUMPHREY, ROBERT A. QUIN.

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 H. A. BUEHLER, R. R. S. PARSONS, HERBERT A. WHEELER.

Chicago

ROBERT W. HUNT, *Chairman*, J. A. EDE, *Vice-Chairman*.
 HENRY W. NICHOLS, *Secretary-Treasurer*, Field Museum of Natural History, Chicago, Ill.
 F. K. COPELAND, G. M. DAVIDSON.

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R. C. GEMMELL, *Chairman*, C. W. WHITLEY, *Vice-Chairman*.
 ERNEST GAYFORD, *Secretary-Treasurer*, 159 Pierpont Ave., Salt Lake City, Utah.
 WALTER FITCH, WILLIAM WRAITH.

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RICHARD MOLDENKE,
SEELEY W. MUDD,
R. V. NORRIS,
EDWARD W. PARKER,
EDWARD D. PETERS,
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ALBERT SAUVEUR,
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RALPH H. SWEETSER,
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HEINRICH RIES,
HENRY L. SMYTH,
FRANCIS A. THOMSON,
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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close April 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Recent Advances in the Chemistry of the Cyanogen Compounds

BY J. E. CLENNELL,* B. S., OAKLAND, CAL.

(New York Meeting, February, 1916)

It is a common observation that the improvements introduced in practice since the first announcement of the cyanide process have been almost entirely mechanical. Although a good deal of study and research has been devoted to the chemical problems involved, the results obtained appear trifling in comparison. Few of the suggested modifications in the chemical treatment have come into practical use and those only in limited fields. Nevertheless, if we study the recent history of the cyanogen compounds from a somewhat broader standpoint, we shall find that considerable additions have been made, not only to our knowledge of their chemical properties, but also to their applications for industrial purposes.

The present review of these advances naturally falls into two divisions: (1) processes involved in the treatment of ores; (2) manufacture of cyanides and other related compounds.

PART I. ADVANCES IN THE CHEMISTRY OF ORE TREATMENT BY CYANIDE

In the attempt to extend the scope of the process, various ores have been encountered which contain gold and silver in such forms that they will not yield satisfactorily to ordinary methods of cyanide treatment. In some cases the difficulty is due to the excessive action on the solution of base metals present in the ore; in others, to the occurrence of minerals such as dyscrasite (the native alloy of antimony and silver), or a supposed oxidized manganese compound containing silver, which are either altogether insoluble in or very slightly attacked by cyanide. A particularly troublesome case is that of the so-called "graphitic" ores, containing carbon in some form which has the property of precipitating gold and silver already dissolved.

Other efforts have been directed to improving extraction by electrolysis of working solutions, and by addition of various chemicals with or without electrolysis; to the diminution of cyanide consumption by "re-

* Metallurgical Chemist.

generating" the spent solution by chemical or electrical means with the object of decomposing complex cyanides formed in the treatment, thus reproducing simple cyanides or other substances capable of dissolving further quantities of precious metal.

Treatment of Carboniferous Ores and Products

Perhaps the most important recent papers bearing on the chemistry of ore treatment by cyanide are those of Morris Green¹ and W. R. Feldtmann.² (The references are to the bibliography at the end of this paper.) Green established the fact that the precipitation of gold by carbonaceous matter contained in certain ores and metallurgical products could be traced to the presence of occluded gases, of which carbon monoxide appears to be the most active. He also pointed out that graphite, anthracite and other dense forms of carbon, natural or artificial, have little or no precipitating effect. It was also found that charcoal gradually loses its precipitating power on exposure to air, and also loses it almost completely when heated to 500° C., if the occluded gases be extracted by a powerful vacuum pump. Feldtmann has carried this investigation a step further, and thrown light on the reaction by which gold is deposited from solutions of its double cyanides by carbonaceous matter.

An investigation was made on a West African ore containing graphitic schist, in the treatment of which the phenomenon not very happily termed "re-precipitation" had been a cause of poor extraction. Unsuccessful efforts were made to remedy the trouble by preliminary treatment of the ore with various metallic salts, oxidizing and reducing agents. Similar substances were added to the cyanide solution itself, but without effect. It was soon observed that an analogy existed between the effects of this graphitic schist, and those of charcoal on gold-bearing cyanide solutions, and that the action was entirely different from that of charcoal on gold chloride solutions. Gold deposited on charcoal from a chloride solution is visibly metallic, and readily soluble in cyanide, whereas gold precipitated by charcoal or graphitic schist from a cyanide solution is almost insoluble in fresh cyanide, and microscopic examination fails to reveal any metallic particles.

Finally it was discovered that a partial extraction of the precipitated gold could be made by treating the impregnated material with solutions of alkaline sulphides. Hydrosulphides of the type NaHS or KHS were the most effective; normal sulphides, as Na₂S, acted nearly as well; polysulphides were less effective and ammonium sulphide still less. It was shown that the gold so extracted appears in solution as an aurocyanide and not as an aurosulphide. It can be readily recovered by precipitation on metallic copper, and also, but apparently with more difficulty, by precipitation with zinc or aluminum.

Feldtmann offers the surmise that the gold compound formed both in

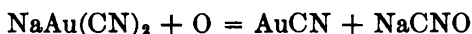
charcoal and in the graphitic schist consists of a carbonyl aurocyanide, possibly $\text{AuCN}\cdot\text{CO}(\text{CN})_2$, insoluble in cyanide but soluble in alkaline sulphides with formation of thiocyanates and aurocyanides, presumably by such a reaction as the following:



An observation which may have an important bearing on the subject is that cyanogen gas is readily occluded by charcoal.

It is to be regretted that the investigation did not also cover the reactions of silver double cyanides under similar conditions. Ores are known in which gold is readily soluble in cyanide whereas the associated silver is extracted very imperfectly. Experiments of R. K. Cowles³ indicate that the cyanide compounds of the two metals show some difference in their behavior in contact with carbonaceous matter. A selective precipitation was observed, first of the gold and later of the silver.

In the discussion on Feldtmann's paper⁴ some alternative theories were put forward. S. J. Speak suggested that charcoal and the schist in question both probably contain oil, which may be the active precipitating agent. H. K. Picard thought the phenomenon might be due to oxidation, by a process similar to that occurring in the decolorization of organic matter by charcoal or platinum black, and stated that cyanates are always found in cyanide solutions after contact with carbonaceous matter of the kind in question. He suggested the reaction:



for the precipitation, and the reaction:



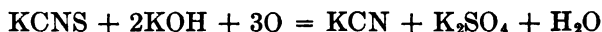
for the re-solution by alkaline sulphide, but admitted that it is difficult to see why, if the gold is precipitated as AuCN , it does not readily dissolve in cyanide. Moreover, if the precipitating effect is due to oxidation, it should be advantageous to pre-treat ores of this class with a strong reducing agent, but Feldtmann's experiments do not bear out this view. Picard, however, pointed out that it would be necessary, after such pretreatment, to prevent the carbonaceous matter from coming in contact with air before the application of cyanide solution.

Several speakers threw doubt on the actual occurrence of graphite in the West African schist. Trewartha-James doubted the theory of adsorbed gases and explained the phenomenon by "surface energy which produces precipitation by purely physical means." S. J. Truscott pointed out that in certain Westralian ores roasting did not completely eliminate the carbonaceous matter and that precipitation effects were still observed in the treatment of the roasted ore. He drew an analogy between this case and that of the manganiferous silver ores refractory to cyanide,

in which pretreatment with a reducer has sometimes proved beneficial. F. P. Menzell thought that the association of this precipitation phenomenon with carbon was purely accidental and considered that other porous material might be expected to act similarly, the chemical composition of the porous medium having nothing to do with the effect.

Electrolytic Regeneration of Cyanide

Turning now to another field of investigation, we find that the process introduced a few years ago by J. C. Clancy has stimulated research on the effects of electrolysis on cyanide solutions, though as yet the results obtained have not yielded any important practical benefits in ore treatment. The Clancy process mainly depends on the electrolysis of water and the secondary reaction of the nascent oxygen thus produced on the thiocyanates present in working solutions, which are converted into cyanides by the reaction:



Unfortunately, when the concentration of KCN in the solution is greater than that of KCNS, the former begins to be attacked, with the formation of cyanate, so that it is rarely possible, even under the most favorable circumstances, to "regenerate" to cyanide more than half the cyanogen which is present as a thiocyanate. Ferrocyanides are oxidized to some extent, by a similar reaction, but the conversion to cyanide is even less complete.

Other difficulties arise in connection with the material to be used as an anode in the process; all ordinary metals are rapidly corroded by the violent action of nascent oxygen at high current density. Peroxidized lead, "passive" iron, graphite, and fused magnetite were found to be the most resistant materials, though even these generally disintegrate in continued use.

In the course of some studies on the electrolysis of aqueous solutions of alkaline cyanides, E. F. Kern⁵ points out that a consumption of cyanide occurs when aqueous solutions are electrolyzed by direct current with insoluble anodes, the consumption being due to oxidation. Anodes of iron, nickel, and lead are dissolved and immediately precipitated, lead as hydroxide and the other metals as mixtures of hydroxides and cyanogen compounds. The consumption of cyanide increases with diminished current density; at a high current density oxygen is evolved and less metal dissolved. Anodes of peroxidized lead and "passive" iron are more permanent than those of pure metals, and are not corroded unless exposed to the air, passive iron being superior to peroxidized lead.

In the electrolysis of cyanide solutions used for leaching refractory

gold and silver ores containing sulphides, no reduced cyanide consumption and no increased extraction were observed, as compared with the treatment of the same ores without electrolysis. I have had a similar experience. When a low current density at anode and cathode was used, some active solvent was produced, but not enough to compensate for loss in electrolysis. The lower the current density at the anode and the higher at the cathode, the greater was the relative cyanide consumption.

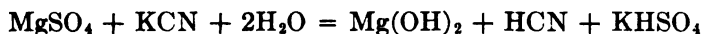
When solutions containing thiocyanates or ferrocyanides were electrolyzed, the loss of cyanide was greatly diminished. Increasing alkalinity also reduced loss of cyanide by increasing conductivity. Peroxidized lead and passive iron anodes were corroded in presence of thiocyanates at current densities above 32 amperes per square meter.

G. H. Clevenger and M. L. Hall⁶ also find that most of the decomposition of cyanide during electrolysis is due to oxygen liberated at the anode by decomposition of water. The final reaction results in the formation of carbonates, and when calcium salts are present, as is usually the case in solutions used for ore treatment, there is a precipitation of calcium carbonate. In some cases the latter might actually interfere with extraction by coating the ore particles.

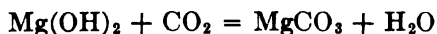
On the whole, the results obtained by different investigators do not encourage the expectations entertained a few years ago that electrolysis will be of material benefit in the treatment of refractory ores by cyanide.

Effect of Mineral Ingredients in Water on Cyanide Consumption

Studies on this subject have recently been made by Thomas B. Stevens and W. S. Bradley⁷ chiefly relating to the cyanicidic effects of calcium and magnesium salts. They find that the common opinion with regard to the action of magnesium salts requires modification. It is known that these salts act as cyanicides by reactions such as



and it is considered that $\text{Mg}(\text{OH})_2$ is useless as protective alkali owing to its insolubility. These writers find, however, that its presence prevents any lime in solution being converted into CaCO_3 by atmospheric or dissolved CO_2 , by reason of the reaction



The $\text{Mg}(\text{OH})_2$ thus acts as a secondary protective alkali by allowing the lime to exert its full efficiency toward cyanicides present in the ore.

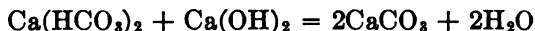
Another point noted by these writers, but not hitherto recognized, is the considerable solubility of calcium sulphate in cyanide solutions. Calcium sulphate and magnesium hydrate, though practically insoluble

in water, are appreciably soluble in the presence of sodium chloride, a matter of importance in the case of the highly saline water sometimes used in Western Australia for making up solutions. It was found that the mixing of comparatively fresh water with solutions made from salt water, and consequently highly charged with calcium and magnesium salts dissolved in NaCl, results in the precipitation of CaSO_4 and $\text{Mg}(\text{OH})_2$. These deposits are injurious, as they coat the zinc and cause bad precipitation of the precious metals; but they are soluble in strong cyanide or in ammonium chloride.

Various salts contained in the water used for making up solutions were found to act as cyanicides, the order of destructiveness being as follows: $\text{Ca}(\text{HCO}_3)_2$, MgSO_4 , CaSO_4 , MgCl_2 , CaCl_2 . The calcium bicarbonate acts much more rapidly than the other compounds. A test made by boiling the water in a reflux condenser, so as to expel CO_2 without reducing volume, and then filtering off the precipitated CaCO_3 , showed that water thus treated had a greatly reduced cyanicide effect. When water containing calcium bicarbonate is used without previously being made alkaline or boiled, the reaction is as follows:



the cyanide acting as a softening agent in a similar manner to lime; compare:



This reaction is the principal cause of the hardening of filter cloths, which occurs when a filter saturated with alkaline solution is washed with "fresh" water containing calcium salts.

The action of magnesium salts is much slower, and results in the gradual precipitation of $\text{Mg}(\text{OH})_2$. Dissolved magnesium salts cannot co-exist with protective alkali even in the presence of much sodium chloride.

PART II. ADVANCES IN THE MANUFACTURE OF CYANOGEN COMPOUNDS

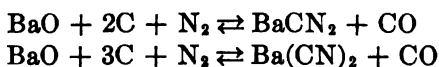
Fixation of Atmospheric Nitrogen

Much activity has been shown of late in the study of methods for the fixation of atmospheric nitrogen, and a number of products are now made on a large scale both for direct use and as a basis for the production of other nitrogenous substances. The most remarkable development has been in the manufacture of calcium cyanamide, CaCN_2 . Impure commercial products consisting mainly of this substance are now extensively used as fertilizers under the names of "lime nitrogen" and "nitrolim." Alkali cyanides and other cyanogen compounds are obtained by further treatment of cyanamides in a variety of ways. Other

processes are used to transform cyanamide into ammonia, ammonium salts, and other nitrogenous compounds of commercial value.

The earlier attempts at the fixation of atmospheric nitrogen were mostly based on the reactions taking place between barium oxide, carbon, and nitrogen at high temperatures. An important investigation of this process has been published by Thomas Ewan and Thomas Napier,⁸ who summarize the chief data obtained by previous investigators in this field. The formation of cyanides from a mixture of BaO, C, and N requires a minimum temperature of 1,200° C., but proceeds best at 1,400° C., when 40 per cent. of the Ba may be cyanized. Hempel (1890) showed that the percentage of Ba cyanized may be increased by pressure. Readman (1894) applied the electric furnace for this process, which has been carried out on a large scale by the Scottish Cyanide Co. Frank and Caro showed (1898) that a large part of the nitrogen is fixed in the form of barium cyanamide, BaCN₂. Caro also found (1907) that the addition of alkali and alkaline earth fluorides accelerates the reaction, and enables it to proceed at temperatures of 900° to 1,100° C. Snodgrass (1907) showed that potassium carbonate has a similar action.

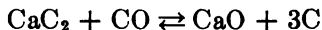
Ewan and Napier (1909) studied the action of catalysts to obtain the reaction at lower temperatures. In their first experiments, mixtures of barium carbonate and charcoal were placed in an iron boat in a porcelain tube, and heated in a current of dry nitrogen. The products finally obtained were barium cyanide Ba(CN)₂, barium cyanamide BaCN₂, carbon monoxide CO, and unaltered BaCO₃, C, and N. The following reactions are supposed to occur:



They summarize their results as follows:

- (1) Absorption of nitrogen begins at 900° to 930° C.
 - (2) The amount absorbed increases rapidly from 1 per cent. Ba combined at 930° to 40 per cent. at 1,000°, in tests made under the conditions that four molecules of N act on one molecule of BaCO₃ for a period of 2 hr.
 - (3) The greater part of the nitrogen is fixed as cyanide Ba(CN)₂ and not as cyanamide. When calcium is used in place of barium, the nitrogen is fixed almost exclusively as cyanamide, CaCN₂.
 - (4) At 960° about 2½ per cent. of the nitrogen is fixed; at 1,000° about 10 per cent.
 - (5) Addition of K₂CO₃ seems to improve the result, but the difference obtained falls within the limits of error in estimating temperatures.
 - (6) No cyanide is produced until about 30 per cent. of the BaCO₃ is converted into BaO, and until the percentage of CO has fallen to 30.
- It is commonly supposed that barium carbide (BaC₂) is formed as

an intermediate product, but Ewan and Napier find no evidence of this. From theoretical considerations BaC_2 could not be formed at all in presence of mixtures of CO and N from which N is freely absorbed by BaO and C. It is probable, however, that when calcium is used instead of barium the reaction:

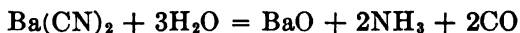


is a necessary intermediary.

Cyanid Gesellschaft⁹ reports that nitrogen has no action on a mixture of CaO and C at $1,100^\circ$, and that the reaction $\text{CaO} + 2\text{C} + \text{N}_2 = \text{CaCN}_2 + \text{CO}$ takes place best at $2,000^\circ$. At this temperature nearly all the nitrogen is absorbed.

K. Kaiser¹⁰ observes that if barium oxide or carbonate, intimately mixed with carbon, be heated to 900° to $1,400^\circ$ C. and allowed to cool in a dry atmosphere of nitrogen under pressure, as much as 90 to 95 per cent. of the theoretical amount of nitrogen combines with the barium.

A French patent of 1913¹¹ describes a method of carrying out the reaction between barium carbonate, carbon, and nitrogen in an electric furnace. Tar is used as binding material for the ingredients; these are molded into balls and part of the tar recovered by distillation. The residue is then heated to $1,600^\circ$ C. in the electric furnace and acted on by "producer gas" or other gas containing nitrogen, to form barium cyanide. The electrodes consist of two rings of carbon placed at the ends of the middle reaction zone of an inclined cylindrical furnace; the fused mass serves to conduct the current. Barium oxide may be regenerated by decomposing the barium cyanide by means of steam:



When calcium is used instead of barium as the vehicle for the fixation of nitrogen, it was found by Barzano and Zanardo¹² that the yield of calcium cyanamide is increased by mixing calcium carbide with sufficient calcium cyanamide from a previous operation to form a friable mass, before subjecting it to the action of nitrogen.

The manufacture of cyanamide by the process of Frank and Caro is carried out on a large scale at the works of the American Cyanamid Co., Niagara Falls, Ont. Nitrogen is obtained by passing air over heated copper. The copper oxide thus formed is again reduced by the action of natural gas, so that it may be used repeatedly. Operations began in January, 1910, with a 10,000-ton plant, which has since been largely extended.

Cyanides and cyanamides of the alkali metals may be produced without the preliminary formation of a calcium or barium compound. E. A. Ashcroft¹³ proposes to do this by using an alloy of the alkali metal with some heavy metal such as lead, and first fusing this with a portion of the

product sought (e.g., NaCN or Na₂CN₂) at about 600° C., to separate the alkali from the heavy metal. The resulting melt, freed from the heavy metal, is now caused to react with cyanogen or a substance yielding cyanogen, with or without electrolysis of the melt.

Conversion of Cyanamides into Cyanides, Etc.

We must now mention some of the recently proposed methods for converting calcium cyanamide into cyanides or other nitrogenous substances of commercial value. These may be roughly classified as fusion methods and wet methods.

Among the former, probably the best known is that of Erlwein and Frank¹⁴ by which the crude calcium cyanamide is converted into a product containing the equivalent of 25 to 30 per cent. KCN, by fusing with carbon and common salt. Careful regulation of the temperature appears to be necessary. I have found that the process is not easily imitated in laboratory experiments. The product has been placed on the market under the name of "surrogat." For extraction of precious metals, it appears to be as effective, per unit of cyanogen contained, as ordinary sodium cyanide. It requires, however, to be digested with water and the insoluble carbonaceous residue filtered off before use.

J. C. Clancy¹⁵ proposes to heat calcium cyanamide with its own weight of a mixture of equal parts of sodium sulphide and chloride in presence of carbonaceous matter.

E. E. Naef¹⁶ treats crude calcium cyanamide at a temperature of 300° to 500° C. with sulphur or a substance yielding sulphur:



The product is then stirred with water and filtered from the residue of CaS + C. The filtrate is boiled gently with zinc dust, and the zinc cyanide formed converted into alkali cyanide by known methods.

Calcium cyanamide may be used as a source of ammonia by acting on it with steam under pressure at a temperature of 170° C., using an agitator to prevent formation of lumps. C. Manuelli¹⁷ observes that under favorable conditions nearly theoretical yields of ammonia can be obtained at a cost of about 1.7c. per pound of nitrogen utilized. Another method (E. E. Naef)¹⁸ is to pass a current of dry hydrogen, either alone or mixed with CO₂, CO, or N, at ordinary or higher pressure, over a heated mass of calcium cyanamide.

The wet methods are based on the fact that cyanamide compounds, such as CaCN₂, are transformed by water into the sparingly soluble crystallizable salt dicyandiamide (CN·NH₂)₂. At a temperature of 13° to 14° C. about 90 per cent. of the nitrogen of calcium cyanamide may be dissolved in 25 times its weight of water, in 8 hr., the extraction proceed-

ing rapidly at first, but very slowly later (C. Manuelli)¹⁷. Various salts of dicyandiamide may be prepared by digesting the substance in presence of acids with the oxides of the metals whose salts are required. The polymerization of cyanamide in aqueous solution is promoted by addition of fixed alkalies or ammonia. According to Grube and Kruger,¹⁸ the reaction appears to be due to the union of undissociated cyanamide ($\text{CN}\cdot\text{NH}_2$) with cyanamide ions ($\text{CN}\cdot\text{NH}$)', to form monobasic dicyandiamide ions $\text{H}\cdot(\text{CNNH})'_2$, presumably in the following stages:

(a) Formation of cyanamide by hydrolysis of calcium cyanamide:



(b) Dissociation of a part of the cyanamide:



(c) Combination of dissociated with undissociated cyanamide:



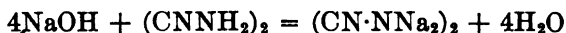
(d) Formation of dicyandiamide:



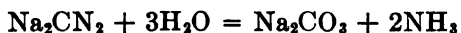
It is found that at a given concentration of total cyanamide, polymerization proceeds most rapidly when the concentrations of cyanamide and of dissociated cyanamide ions are equal.

For the preparation of dicyandiamide from commercial calcium cyanamide, the addition of a foreign base is unnecessary; the conversion may be brought about by precipitating the lime at intervals so as to maintain approximately equal concentrations of $\text{CN}\cdot\text{NH}_2$ and $(\text{CNNH})'$.

Alkali salts of dicyandiamide are produced by heating the substance with caustic alkalies in presence of an absorbent of water:²⁰



If the water is not absorbed, part of it reacts with the melt thus:



The absorbents used are alkali or alkaline earth metals or their oxides, amides, nitrides, carbides or alloys.

Dicyandiamide is converted into cyanide by fusing with alkaline carbonates and carbon:



or it may be converted into a variety of nitrogenous products suitable for manures, explosives, etc.

The conversion of cyanamide into dicyandiamide may also be brought about by heating its solution with a catalytic agent. H.

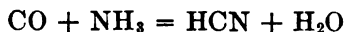
Immendorf and H. Kappen²¹ propose to carry out this process by means of ferric oxide or hydrate, manganous or manganic oxides or their hydrates, stannic acid, chromic oxide or hydrate, hydrated silicic acid or similar substances. The products obtained are ammonia, urea, and dicyandiamide. During the process the catalyst absorbs the bases liberated and loses its efficiency. This may be restored by treatment with acid or with water. Commercial calcium cyanamide contains calcium chloride, which hinders the separation of urea and dicyandiamide. This difficulty may be avoided by converting the calcium chloride into sulphate or carbonate, which may be separated by reason of their insolubility in alcohol.

H. Kappen²² obtains thiourea by treating dilute aqueous solutions of cyanamide with hydrogen sulphide at high temperatures, in closed vessels under pressure. The addition of acid and alkaline reagents and of soluble compounds of arsenic, antimony, or tin increases the efficiency of the process.

Production of Cyanogen Compounds from Nitrogenous Gases by Catalysis and Otherwise

Another method for the manufacture of cyanides and related compounds, which has been largely developed of late years, consists in the combination, under suitable conditions, of volatile carbon and nitrogen compounds. Usually this takes place at a high temperature under the influence of a "catalyst," i.e., of some substance which does not undergo any permanent chemical change in the process.

It has long been known that hydrocyanic acid or ammonium cyanide could be produced by passing mixtures of carboniferous and nitrogenous gases over heated platinized pumice, as for example in Woltereck's process:²³



C. Beindl²⁴ proposes to form hydrocyanic acid and cyanogen compounds by catalytic combination of gaseous and volatile carbon compounds with gaseous and volatile nitrogen compounds, by the use of a contact material consisting of a metallic oxide or a mixture of metallic oxides. The gaseous mixture should contain not more than 7 volumes of the carbon compound to 1 volume of the nitrogen compound. It is claimed that oxides of metals of the iron group have the property of inducing the rapid formation of hydrocyanic acid at relatively low temperatures, from dry mixtures of these gases.

It has been found²⁵ that the yield is increased by introducing into the arc of an electric furnace the vapors of metals and metallic compounds, such as those of copper or iron and their salts.

W. Moldenhauer and O. Wehrheim²⁶ describe a method by which

nitric oxide can be obtained by catalytic combustion of cyanogen compounds. Ammonia may also be used as the source of nitrogen, but the yield of nitric oxide is less than with cyanogen, and more attention must be paid to the condition of the catalyst and time of contact with it.

J. E. Bucher²⁷ proposes to produce alkali cyanides by the action of atmospheric nitrogen on alkali carbonate and carbon dissolved in iron, the iron acting as a catalytic agent. The alkali cyanide formed is removed from the reaction zone by distillation under diminished pressure.

The Deutsche Gold und Silber Scheide Anstalt²⁸ propose to pass gaseous mixtures containing nitrogen obtained by heating certain waste products such as "vinasses" through highly heated conduits lined with refractory material such as Dinas brick, fused quartz, or a fused mixture of zirconia and quartz which remains gas-tight and non-porous at the high temperature employed. It is said that practically the whole of the nitrogen is obtained in the forms of HCN and NH₃, the relative proportions of these varying with the raw material used.

In the process of Barzano and Zanardo²⁹ a mixture of nitrogen (say 60 per cent.), hydrogen (32 per cent.), and hydrocarbons (6 per cent.) is raised to a high temperature, for example in the electric arc, and hydrocyanic acid absorbed by circulating the issuing gases through cooling and absorption apparatus. Sufficient nitrogen is added to maintain, with the hydrogen liberated, the concentration of these gases most favorable to the reaction.

Catalysis may be employed for the production of ammonia, not only from gaseous mixtures but also from solid organic nitrogenous compounds. F. Schreiber³⁰ proposes to act on the latter with contact masses containing hydrated iron oxide at temperatures below red heat. It is claimed that as much as 80 per cent. of the nitrogen in pyridine and in cyanogen salts may be thus converted into ammonia, while ordinary destructive distillation yields not more than 20 per cent.

Production of Cyanogen Compounds from Gaseous Mixtures by Absorption with a Metallic Salt

The cyanogen contained in crude illuminating gas has for some years been utilized as a source of cyanides and cyanogen compounds on a commercial scale, as in the methods of Rowland³¹ and Bueb.³²

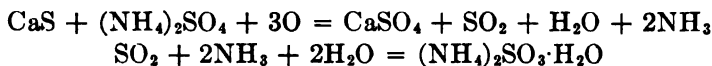
L. Bergfeld³³ uses as purifying material dehydrated copper sulphate, or a mixture of salts, *e.g.*, alkaline earth sulphates or chlorides, which combine with NH₃ but not with H₂S, with metallic oxides capable of combining with H₂S. The mixture is treated with ammonia gas before use. The reaction is of the type:



At the same time some SO_2 (which is fixed by the purifying material) and sulphur are also formed, thus:



When the purifying material is exhausted it is heated, with regulated access of air, and ammonia liberated as follows:



By gentle warming in presence of air, the ammonium sulphite deposited is rapidly oxidized to sulphate.

Cyanogen in the crude gas is fixed by the purifying material as cuprous ammonium cyanide. On heating with air this yields NH_3 and CO_2 , any thiocyanate giving NH_3 , CO_2 and SO_2 . After passing the purifying material the coal gas may be further purified by leading it over alkali or alkaline earth sulphides, and if necessary over sodium amalgam.

C. A. Bergh³⁴ uses a solution of a zinc salt, containing also ferrous chloride, as an absorbent. The free acid formed is neutralized, and zinc sulphide and cyanide are precipitated, by addition of calcium carbonate, or some other sparingly soluble substance. This is added in quantity corresponding to the amount of H_2S and CN present. Iron remains in solution. By this method zinc is separated from iron and H_2S from CN by a single operation.

W. H. Coleman³⁵ passes coke-oven or other gases containing hydrocyanic acid through a tower packed with lumps of ferrous iron ore. A solution of sodium carbonate, or other suitable alkaline absorbent in solution or suspension, is circulated through the tower, and the resulting alkali or alkaline-earth ferrocyanide liquor is treated for recovery of the ferrocyanide.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

The Control of Chill in Cast Iron. Considering the Elements Effective in the Manufacture of Malleable Castings and Chilled Car Wheels

BY GRAFTON M. THRASHER,* MICHIGAN CITY, IND.

(New York Meeting, February, 1916)

For the proper discussion of this subject it is necessary to incorporate in this paper the substance of part of an article published by me in *Metallurgical and Chemical Engineering* for January, 1915.

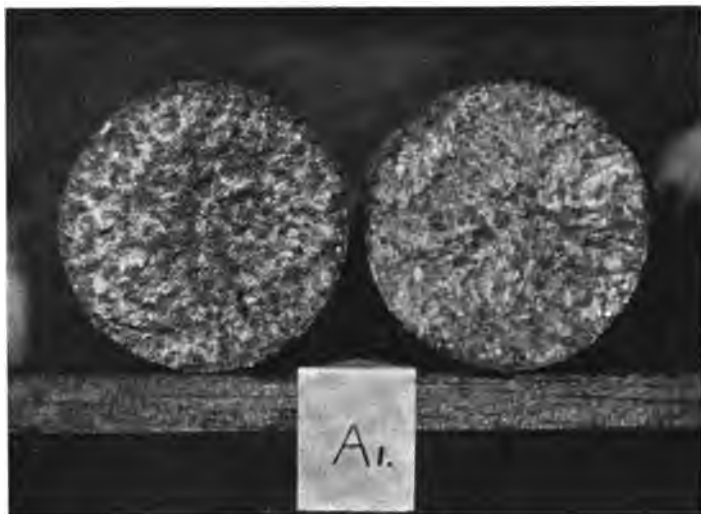


FIG. 1.—CAST-IRON TEST PLUG, SHOWING LOW-MOTTLED AND MOTTLED FRACTURES.

Several years ago, while I was chief chemist for a malleable-iron company, it occurred to me that the mere regulation of silicon was not sufficient to produce the desired degree of chill. But only after repeatedly encountering this fact in both open-hearth and air-furnace, as well as cupola metal, was I led to regard variation in the total carbon as the principal cause of disturbance; and I resolved to determine the total carbon as well as the silicon in every heat, and by a carefully prepared test piece to gauge the approximate degree of chill. The test piece selected was a round plug, 2 in. in diameter and 12 in. long, cast on end

* Chief Chemist and Metallurgist, Haskell & Barker Car Co.

in green sand. It was allowed to cool at least 2 hr. in the mold before being cooled in water and broken in the middle; and the fractures were designated as gray, low mottled, mottled, high mottled, and white. All these fractures except the well-known gray one are illustrated in Figs. 1 and 2. From the same ladle of iron from which the test piece was poured a shot sample was taken for analysis so as to represent the actual iron in the bar as near as possible; and the test in all cases was poured from iron taken from the furnace when about half emptied.

It may be well to mention here that the principal disturbing influence on these tests is the fact (proved by experiment) that it is entirely possible to pour a mottled test and a gray test from the same ladle of iron, depending upon the pouring temperature.

In order to study the cause of the variations more systematically a

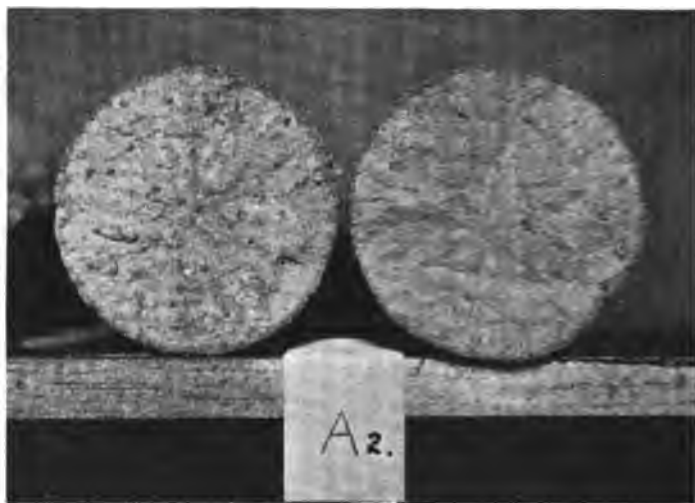


FIG. 2.—CAST-IRON TEST PLUG, SHOWING HIGH-MOTTLED AND WHITE FRACTURES.

chart was prepared for plotting the analyses with carbon as ordinates and silicon as abscissæ. It soon became apparent that the analyses corresponding to the test piece with average chill, designated as "mottled," were arranged along a diagonal line on the chart, with analyses corresponding to high and low chill at a more or less definite distance on either side of the line. It happened that about that time a heat had to be held pretty long in the furnace on account of a breakdown, and it became necessary to doctor it up with 50 per cent. ferro-silicon to get the metal into pouring condition. The test piece from this heat was found to be just normal or what is designated above as "mottled," and the corresponding analysis was silicon 1.07 and carbon 2.47 per cent., which served admirably for one extreme of the chill line as drawn on the chart. Something to represent the other extreme was found in a

pig of charcoal iron of the same dimensions as the test piece and having the proper degree of chill.

More recently I have obtained data for a new high-silicon point in which the analysis was, silicon 1.25, carbon 2.25 per cent.

The chart as finally constructed is shown in Fig. 3.

As the manganese in all of these heats ranged between 0.25 and 0.35 per cent., which is normal for air-furnace malleable, and as sulphur averaged about 0.06 per cent., it will be seen that these elements interfered but little with the results. The phosphorus, of course, was between 0.13 and 0.18 per cent. and likewise had very little influence.

In examining the chart it will be noted that while the line tends

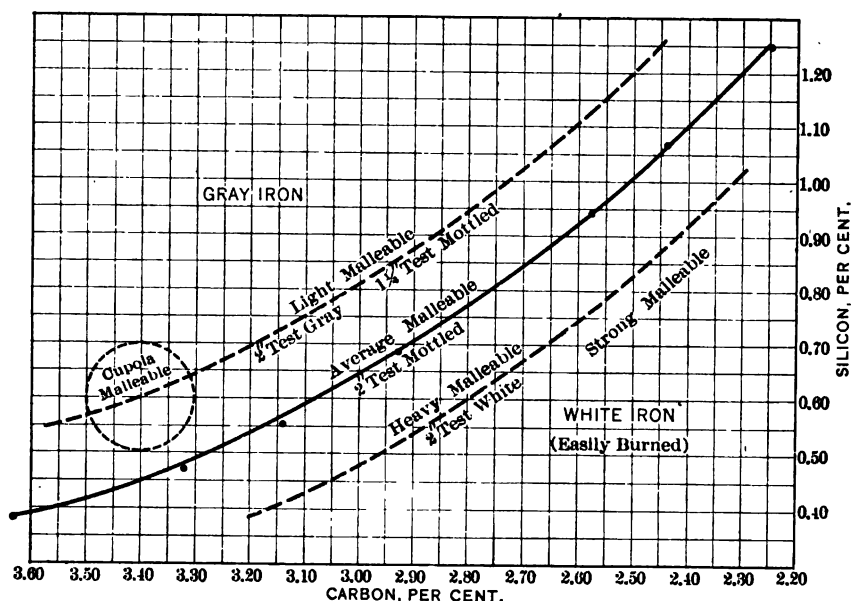


FIG. 3.—LINES OF EQUAL NATURAL CHILL.

to curve somewhat, yet it is sufficiently straight through the usual range of analyses to permit the assumption for practical purposes that 4 points of carbon are equivalent to 3 points of silicon in the regulation of chill, when the silicon is above 0.70 per cent.; and that below this about 2 points carbon are equivalent to 1 point silicon.

The value of this knowledge will be at once apparent to a malleable-iron foundryman, as it enables him to substitute large quantities of low-carbon material for his sprue in case of shortage of that material, thus:

It is desired to substitute steel containing silicon 0.10 and carbon 0.30 per cent. for 1,000 lb. of sprue containing silicon 0.80 and carbon 2.80 per cent.

Equivalent deficiency of silicon in the steel

due to low carbon = $(2.80 - 0.30) \times \frac{3}{4} \times$

1,000 = 18.7 lb.

Silicon deficiency = $(0.80 - 0.10 \times 1,000 = 7.0$

Total equivalent silicon deficiency..... 25.7

Therefore, it is necessary to increase the silicon in the mixture 25.7 lb. by substituting higher-silicon pig iron.

The popular conception of malleable iron accepts the limits of 0.60 to 0.90 per cent. in silicon, with a very indefinite notion as to what the carbon content in the hard state should be; but a careful examination of this chart will show that for a given silicon percentage there is a definite carbon percentage, which is proper for each class of castings.

In order to prevent cracked work and for other economic reasons, the malleable-iron foundryman runs his mixture as nearly gray as he dares, but if he desires to produce a very strong iron the presence of primary graphite in the hard iron must be avoided by lowering either the silicon or the carbon or both.

As to malleable iron produced with the same average chill but with varying analyses, it will be found that the high-silicon, low-carbon mixture is more sound and reliable, but possibly a little less fluid, while the high-carbon, low-silicon mixture is subject to greater shrinkage and segregation, which produce weakness.

The principal consideration for the foundryman is to produce that type of structure which is necessary to obtain malleability upon annealing; and the chart shows the proper adjustment of silicon and carbon to bring about this desired result for the different grades of work.

It will be further noted that, of the three factors considered, namely, silicon and carbon content, and chill, if any two be known the other may be predicted approximately.

Malleable iron produced from the cupola is pretty well confined to the dotted circle shown on the chart. Its proximity to gray iron, as well as its position in the field of greater shrinkage, will account for its weakness, which is not due as popularly supposed to the higher sulphur content. As the carbon is automatically adjusted within this range by melting in the cupola, the regulation of chill must be effected by silicon alone. It is not feasible, however, to run a mixture lower than 0.50 per cent. in silicon because the cutting action of the slag thus produced is very hard on cupola linings. Good malleable castings are produced along the whole length of the charted lines; but the average foundryman, with his limited knowledge of conditions to be met in another foundry, cannot see how the other party makes a certain casting with 0.90 silicon while he finds it necessary to make it with 0.60 silicon. It is hoped that the publication of this chart will clarify the matter. The lines here

given have been confirmed by plotting hundreds of analyses of mixtures of known chill and normal pouring temperature and the chart has been found a valuable aid in practical work.

Regarding the strength of malleable iron produced from irons of varying degree of chill, the following data, charted in Fig. 4, from test bars poured every other day for two weeks, will give a good idea of the effect of this factor:

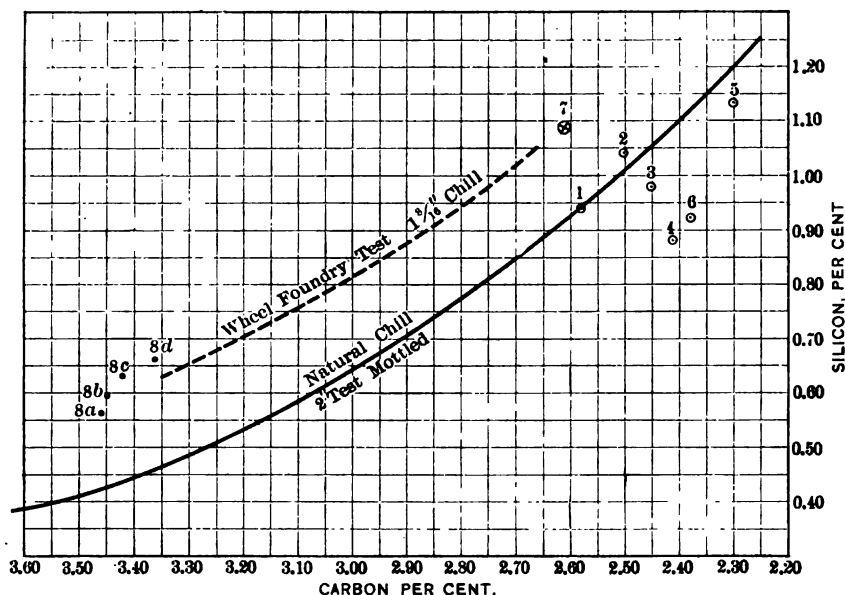


FIG. 4.—ANALYSES OF BARS, TESTED FOR TENSILE STRENGTH.

Bar No.	Tensile Strength, Pounds per Square Inch	Elongation, Per Cent.	Silicon, Per Cent.	Carbon, Per Cent.
1	43,680	10.0	0.94	2.58
2	44,770	9.5	1.04	2.50
3	48,340	10.5	0.98	2.45
4	49,710	13.0	0.88	2.41
5	49,440	12.0	1.03	2.30
6	48,080	11.0	0.92	2.38

These test bars were round, 0.8 in. in diameter for 2 in. in the middle, and enlarged at the ends for gripping. A shrink ball was cast on one end to insure soundness. The structure of two of these bars is shown in Figs. 5 and 6. No. 1, the weaker bar, shows slight inclination to graphitic structure in the shape of the carbon particles.

The nearest approach that I have noticed to the co-ordination of

such valuable facts as are above outlined is the paper published in the *Proceedings of the American Foundrymen's Association*, a couple of years

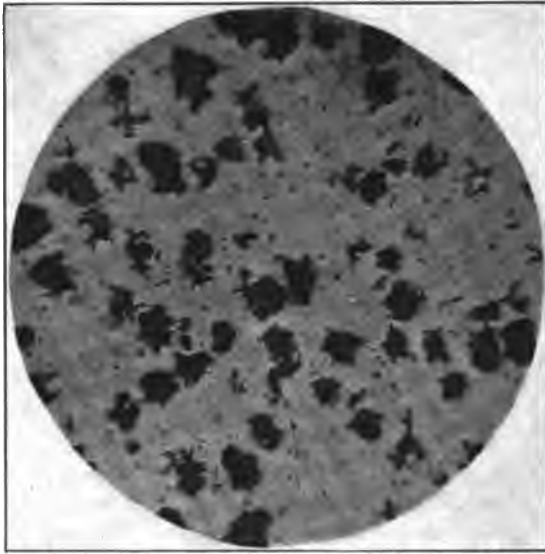


FIG. 5.—STRUCTURE OF BAR NO. 1, MAGNIFIED 50 DIAMETERS.

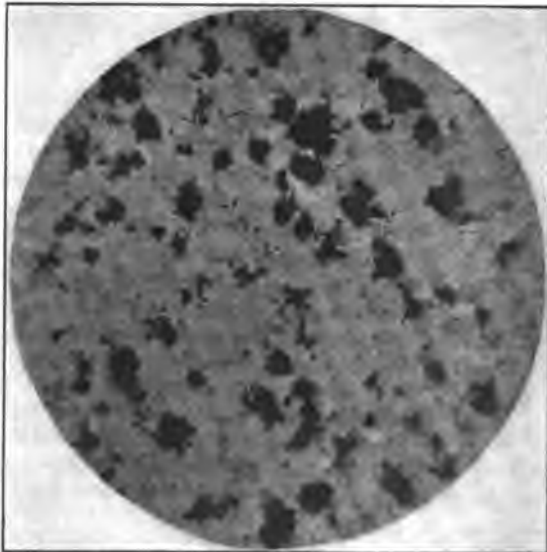


FIG. 6.—STRUCTURE OF BAR NO. 4, MAGNIFIED 50 DIAMETERS.

ago, by John Jermain Porter, in which he plotted silicon and combined carbon; but as he made no statement as to the size of the castings from which these results were obtained, it is probable that this factor

was not kept constant, which of course would account for a great many irregularities. Also, no reference was made to the effect of the total carbon on the combined-carbon content, which, as may be seen above, is highly important.

For the purpose of comparing the "natural chill" with that produced when cast iron is poured against an iron chiller, a test was made which conformed to that used in a car-wheel foundry, using the same pattern and chill box. A ladle of very hot low-carbon iron was treated with sufficient 50 per cent. ferro-silicon, as calculated from the chart, to produce the same degree of chill as that of the iron used in car wheels. It was necessary to allow the metal to cool for several minutes in the ladle before pouring, in order that the temperature should approximate the practice in the wheel foundry, where the metal is not so hot when melted. The result of this experiment is charted in Fig. 4 as No. 7, while a regular test piece from a wheel mixture showing the same degree of chill ($1\frac{3}{16}$ in.) is charted as No. 8. The significance of this experiment is that it points the way to the production of car wheels with high silicon, low carbon, and low sulphur by the use of an open-hearth furnace for melting. The rapidly increasing sulphur in the cast-iron car wheel of today, due to the remelting of old wheels, is a source of anxiety to both the manufacturer and the railroad. The percentage of cementite in the structure of such a metal would still be sufficient to insure good wearing qualities and a higher percentage of silicon would insure soundness. The absence of large quantities of manganese sulphide, which is undoubtedly the cause of a large percentage of the defects in the tread, would be a strong argument in favor of such a metal.

It may be profitable to consider here the relationship of the "lines of natural chill" as described above to the line of saturation of silicon and carbon in cast irons. For this purpose I have prepared another chart, Fig. 7, which outlines this relationship. The saturation figures are taken from Keep's *Cast Iron*, p. 45, and from the work of Wüst and Petersen, as outlined by Hatfield.¹ The line I produced, which is that of "natural chill," is the same as that marked "2" Test Mottled" in Fig. 3. As a speculation it has been extended toward 6.67 carbon, as its probable trend in that direction, and upward to 3.50 silicon and 1.25 carbon, where it most probably tapers off and disappears due to the absence of carbon in sufficient excess of the pearlitic ratio to form graphite, which is necessary to produce the mottled fracture described. I hoped to produce by direct experiment a more definite end point on the high-silicon side before presenting this paper; but lack of time prevented.

It may be well to add a word concerning the influence of the other elements necessarily present in commercial irons which have been disregarded in this paper.

¹ *Cast Iron in the Light of Recent Research*, p. 31 (1912).

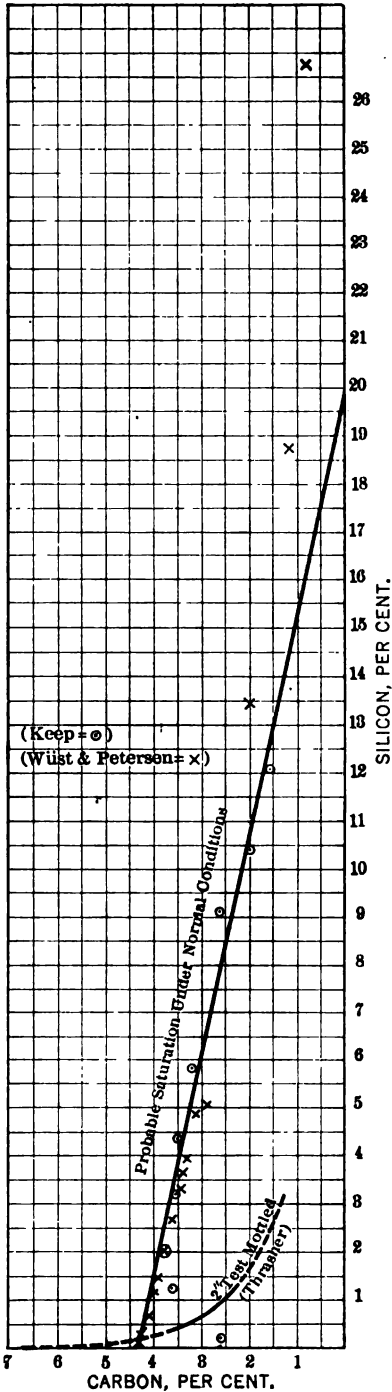


FIG. 7.—RELATION OF LINE OF NATURAL CHILL TO THAT OF SATURATION.

The effects of manganese and sulphur must be considered together, since it has been shown by several eminent investigators that where present together in the proper ratio they exist as MnS , which, properly distributed, would be neutral and ineffective upon the chill or strength of the metal. In practical work it is found necessary to carry an excess of manganese above this ratio, to insure reliable metal. In cupola-melted irons where the carbon is high, an excess of from 0.05 to 0.10 per cent. of manganese is sufficient, while iron melted in the air furnace or open hearth requires from 0.10 to 0.30 per cent. excess on account of greater exposure to oxidizing influences. It will be seen that where iron is properly melted to a fluid condition, the sulphur is fixed and without

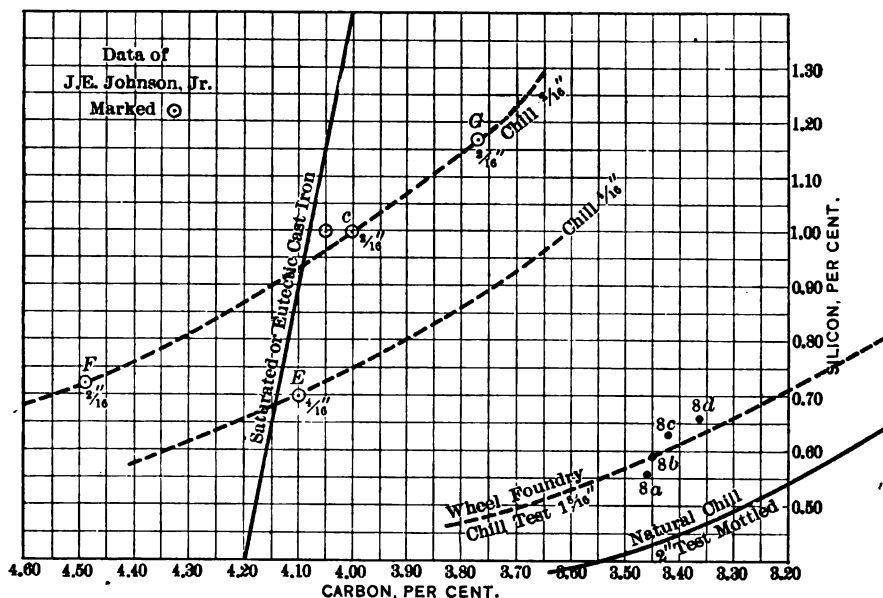


FIG. 8.—DATA OF J. E. JOHNSON, JR.

influence on chill, etc. If the 0.30 per cent. excess manganese is figured to Mn_3C it will be apparent that it can affect only 0.02 per cent. of carbon, which would have but small influence on the chill.

In my opinion the effect of sulphur on the chill, even when present uncombined with manganese, has been greatly exaggerated, and oxygen occurring simultaneously will be found responsible for this exaggeration.

Regarding the effect of phosphorus on the chill, it will be sufficient to cite Hatfield, who says, in discussing Stead's work on this element:

"Since, however, there is seldom more than 1.5 per cent. of phosphorus present in pig irons used in foundry work, it may be considered that the influence of such percentages upon the condition of the carbon is negligible."²

It will be seen that in the control of chill in malleable iron and car wheel mixtures, where this element does not exceed one-fifth of 1.5 per cent., its effect cannot be noticed.

Though I have not made an independent investigation of the effect of oxygen on the chill, I am inclined to believe from the data at hand that its effect is mainly qualitative rather than quantitative. While I fail to agree with some of the conclusions of J. E. Johnson, Jr.,² in his paper on this subject, I think his theory of the existence of an oxy-sulphide is true. In fact, a close analysis of Mr. Johnson's data brings out more forcibly my contention, namely, that the quantitative control of chill in commercial cast irons, aside from the physical effects of casting temperature, size of test, and size of chiller, is practically dependent upon the percentages of silicon and total carbon present. In support of this statement, a chart, Fig. 8, is offered, in which Mr. Johnson's data are tabulated. His *A* and *B* series, not having a measurable chill, have been left out. It will be noticed that his *G*, *C*, and *F* series, forming a line for $\frac{1}{8}$ -in. chill, and his *E* on a line representing $\frac{1}{4}$ -in. chill, are in perfect agreement with my data, shown in the lower right-hand corner, and all lines verge toward the point representing 6.67 carbon and no silicon. While his specimen *D* does not seem to agree with this theory, it is entirely possible that it may not be in its proper position, either because of error in analysis, making silicon or carbon too low, or because the sample was not representative; though a larger test pig than usual, or very low pouring temperature, would cause the specimen to show no chill.

In concluding I wish to express my acknowledgments to members of the Institute, and especially to Professors Howe, Sauveur, and Stoughton, whose work on cast irons has been helpful and inspiring in the study of this interesting subject.

² Hatfield: *Cast Iron in the Light of Recent Research* p. 54 (1912).,

³ *Trans.*, 1, p. 344 (1914).

*** DISCUSSION OF THIS PAPER IS INVITED.** It should preferably be presented in person at the New York meeting, February, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 20 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

The Newnam Hearth

BY WILLIAM E. NEWNAM,* B. S., COLLINSVILLE, ILL.

(New York Meeting, February, 1916)

THE smelting of galena in the ore hearth has been practiced in many countries for several hundred years with varying success. In the United States the water-jacketed American hearths and the Jumbo hearths have found some favor, in the Missouri lead belt, where large quantities of non-argentiferous galena concentrates are produced.

In form and method of operation the ore hearth has changed but little since its earliest conception. A detailed description of the furnace is not needed here; it has been fully described in most works on the metallurgy of lead and in many technical papers. Since its field is restricted to non-argentiferous galena, containing over 68 per cent. of lead, its use is limited to a few districts; and, on account of certain disadvantages attending its operation, modern sintering and blast-furnace practice has nearly driven it out of the Missouri field.

It is not generally known that the cost of producing pig lead is smaller by the hearth method than by sintering and blast-furnace smelting. Yet it is doubtful if any metallurgist knowing this to be true would recommend an American hearth installation—by reason of the disadvantages which will be briefly enumerated.

The furnace has been limited to a length of from 4 to 5 ft., requiring for its operation two men working an 8-hr. shift as a maximum. The product per man is small; the work is hot and laborious; and the difficulty of completely removing the dust and fume has made it a notorious source of lead poisoning. Although suitable for operations on a small scale a large number of hearths would be required for a plant smelting 10,000 tons of 68 to 70 per cent. concentrates per month; and the number of semi-skilled laborers would be six times the number of hearths—a condition which is prohibitive.

On account of the high percentage of dust and fume produced, a large flue and bag-house installation is required and, in the past, the handling and re-treatment of this dust and fume has been a serious problem.

* Superintendent, St. Louis Smelting & Refining Co.

On the other hand, the advantages are the immediate reduction of from 55 to 65 per cent. of the lead contents of the concentrates, with simultaneous roasting, thus producing a small tonnage of gray slag, which is an ideal product for the lead blast furnace, since it greatly accelerates the furnace operation and is productive of but little matte, the lead being largely present in the form of sulphate and oxide. As previously observed, the cost per ton of pig lead is somewhat cheaper by this method.¹

Several attempts have been made to raise the tonnage and reduce the labor on the ore hearths by mechanical means, but up to the present without success. Although the furnace operation appears to be extremely simple, this is really not the case; and the ore hearth has persistently refused to respond to mechanical improvements. It is the purpose of this paper to describe a new and more fortunate endeavor in this direction.

After a careful study of the hand-hearth operation, it was decided that three things were necessary to perfect it, namely: A cool and sanitary hood; a mechanical rabbler which would effectually replace the continuous and laborious use of the hand rabble; and a lead-well attachment that would mold clean lead direct from the hearth basin with little attention on the part of the furnacemen.

All our experiments were carried out on the ordinary 4-ft. hand hearth. The hood problem was taken up first; and soon a double hood was devised which gave the furnace room a clear and fume-free atmosphere, at the same time reducing the direct and radiated heat to the workmen 50 per cent. A complete cessation of sickness and an increased output were at once noted.

After a few trials a simple and effective lead well and molding device was installed.

A traveling rabbler was next put on the 4-ft. furnace, and, after numerous changes, a successful type was worked out.

This rabbling machine, as shown in Figs. 1 and 2, is hung from a carriage which travels on an overhead track; and it rabbles in one direction only, for a reason given later. The machine being in such a position that the rabble arm is at the extreme end of the furnace, a releasing lever is pulled, which starts the machine, causing the rabble arm to describe a motion which is similar to that of the hand rabble, but more effective, since there is more power behind it. As the rabble arm is withdrawn from the fire, an eccentric connected with a ratchet wheel moves the carriage forward about 4 in. for the next stroke of the rabble arm. This motion is repeated until the other end of the furnace is

¹ It may be of interest at this point to note that the lead blast furnace carrying a charge with 20 per cent. of gray slag is smelting 7.5 tons of charge per square foot of tuyere area per 24 hr. The lead in this slag is under 1 per cent. and the proportion of fixed carbon consumed per charge 7.5 per cent.

reached, when the machine automatically stops and withdraws the rabble arm from the fire.

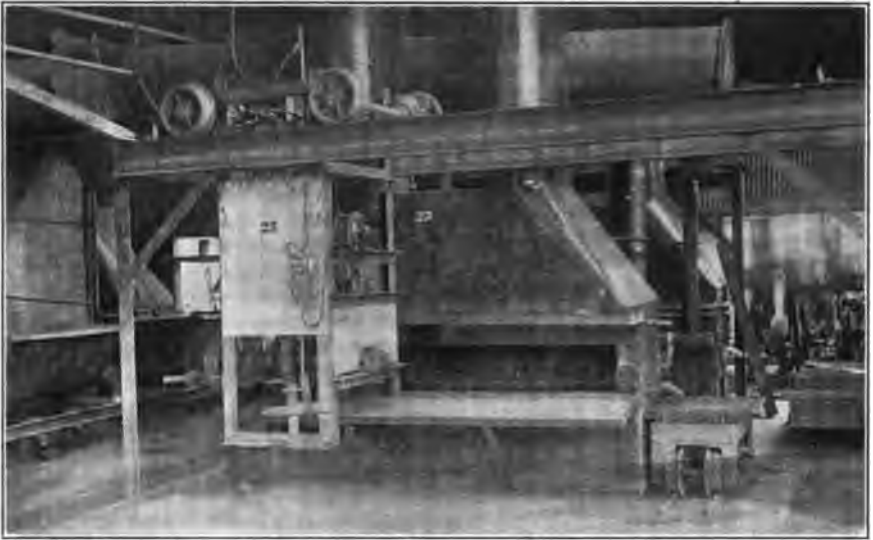


FIG. 1.—RABBLING MACHINE AT BEGINNING OF TRIP.

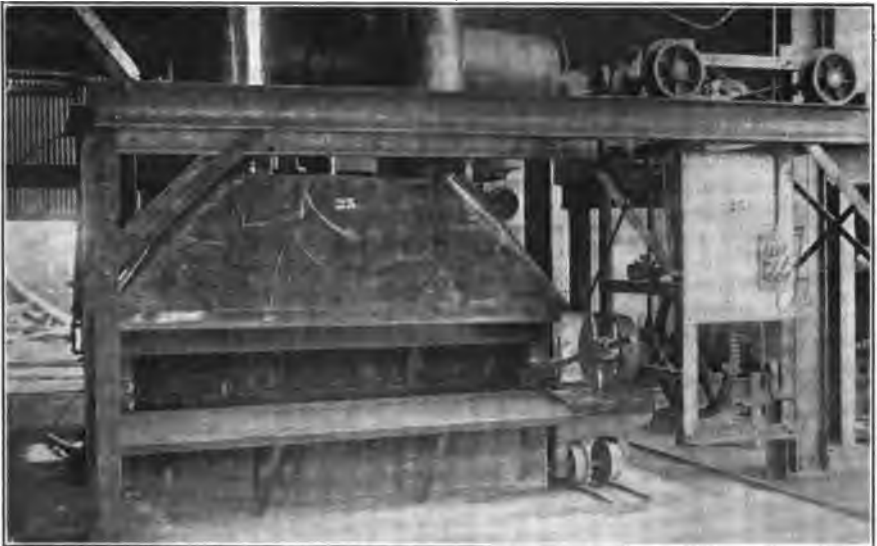


FIG. 2.—RABBLING MACHINE AT END OF TRIP.

Two men, a charger and a helper, operate the furnace. The helper follows close behind the rabble arm and with a long-handled shovel pushes back the loose charge, picking out the gray slag as he goes along. Close

behind the helper comes the charger, who spreads a thin layer of ore on the charge as fast as it is shoveled back by the helper, adding coke breeze as needed..

Thus the fire is exposed for an instant only; and, by the time the trip down the furnace is completed, the end first charged is ready for the rabbling machine. It is for this reason that the rabbling is performed in one direction only. The first trip being complete and the gray slag removed from the apron, the throwing-in of a clutch causes the carriage to return without rabbling to its original position, where it automatically stops in readiness for a second rabbling trip.

Thus on each trip the fire is rabbled 24 times in 48 sec., the return motion requiring 12 sec. One horsepower is required under full load, and is supplied by a 1-hp., alternating-current, 220-volt motor. The rabbling machine is simple, strong, and durable, and only requires a few minutes' attention daily.

From the data compiled it was determined that the hearth could be increased to 8 ft. in length, that it would then produce 2.5 times as much pig lead as the hand hearth, and that two men could operate it with less fatigue than that sustained by two men on the ordinary hand hearth. An 8-ft. hearth was at once constructed, and from the first day of its operation it has not only realized our expectations but considerably exceeded them.

As direct-hearth labor is paid a certain rate per hundred pounds of pig lead produced, it will be seen that the cost of production as well as the number of furnaces and number of men employed has been more than cut in half. Moreover, the hearth laborers can earn more money per shift, through the increased efficiency per square foot of hearth area; and this, in turn, enables us to secure a more desirable class of men.

The following tables show that the 8-ft. furnace gives a much higher extraction of metallic lead, with a corresponding decrease in the amount of dust and fume produced:

Another gratifying feature is the reduction of the fuel added in the form of coke breeze. Whereas the hand hearth uses from 8 to 9 per cent. of fuel, the 8-ft. mechanical hearth consumes from 3 to 4 per cent. only. It is probable that the decrease in dust and fume as well as in fuel consumption is due to the shorter time the fire is exposed after rabbling on the 8-ft. hearth.

It has been found that by pugging the flue dust with a certain cheap chemical, and mixing it with a portion of burnt bag-house fume, the mixture can be successfully worked up on the hearth, giving a high lead extraction and a very low percentage of dust and fume.

Thus it will be seen that for galena concentrates of over 68 per cent. lead contents, the disadvantages of the old-style hearth have been overcome, and the cost of production has been lowered to such a degree that

sintering with subsequent blast-furnace smelting is no longer comparable therewith.

Smelting Flue Dust on the Hearth

The following is an average of the results obtained by smelting a mixture of pugged flue dust and burnt bag-house fume. This mixture is made up so that the average weight of pig lead produced per shift will be approximately the same as that extracted from 70 per cent. galena concentrates.

Owing to the low percentage of sulphur present in the mixture as sulphides, 10 per cent. of coke breeze is required.

Burnt bag-house fume, containing 76 per cent., and pugged flue dust containing 62 per cent., are made into a mixture containing 67 per cent. of lead. Of this lead, 81.3 per cent. passes into pig lead, 10 per cent. into gray slag, and 8.7 per cent. into fume and dust.

The gray slag contains: Pb, 32.4; FeO, 14.7; CaO, 11.1; S, 1.2; and insoluble, 18.7 per cent.

The following table is a comparison of the two types of hearth, operating side by side on the same ore, showing the average results per 8-hr. shift over a period of four weeks; the labor being the same on each type.

Comparative Results

Ore charged, galena concentrates, containing 72.5 per cent. lead, and 15.1 sulphur.

	Newnam Hearth, Pounds	Hand Hearth, Pounds
Dry ore charged.....	13,179	5,091
Lead contents.....	9,554	3,691
Pig lead made.....	6,443	2,030
Gray slag made.....	3,318	1,329
Per cent. of coke breeze used.....	3.6	8.8
Rabble trips per hour.....	37.2	

Percentage of Total Lead in Products

Pig lead.....	67.44	55.00
Gray slag.....	15.18	16.20
Dust and fume.....	17.38	28.80
	100.00	100.00

Analysis of Gray Slag

	Pb	FeO	CaO	S	Insoluble
Newnam hearth.....	43.7	12.8	9.9	1.9	12.6
Hand hearth.....	45.0	12.2	9.0	2.6	12.2

The lead present is approximately combined as follows:

	Per Cent.
PbSO ₄	10
PbS.....	6
PbO.....	25
Metallic lead.....	8.5

From the standpoint of roasting, the elimination of sulphur on the hearth is high in comparison with the other methods now in vogue.

Of the material drawn into the flues and bag chambers the proportion by weight is one of dust to two of burnt fume. From the analyses given below it will be seen that a considerable portion of the lead is present as sulphate, and on re-treatment the sulphate, reacting on the sulphide, greatly assists the desulphurizing process.

	Pb, Per Cent.	S, Per Cent.
Dust.....	62.0	11.1
Burnt fume.....	76.0	5.9

The lead present is combined approximately as follows:

	Dust, Per Cent.	Burnt Fume, Per Cent.
PbSO ₄	18.9	53.6
PbS.....	55.4	1.2
PbO.....	2.0	41.5

Taking the above average run as a basis, the sulphur elimination is calculated as follows. The amount of fume and dust produced from the first dust and fume treatment is so small a percentage of the whole that it will not be considered here.

Newnam Hearth

	Pounds	Sulphur, Per Cent.	Sulphur, Pounds	Sulphur, Pounds
Dry ore charged.....	13,179	15.1		1,990
Gray slag.....	3,318	1.9	63	
Dust.....	776	11.1	86	
Burnt fume.....	1,552	5.9	92	241
Pounds of sulphur eliminated...				1,749
Per cent. of sulphur eliminated..				87.9

Hand Hearth

	Pounds	Sulphur, Per Cent.	Sulphur, Pounds	Sulphur, Pounds
Dry ore charged.....	5,091	15.1		769
Gray slag.....	1,329	2.6	35	
Dust.....	497	11.1	55	
Burnt fume.....	994	5.9	59	149
Pounds of sulphur eliminated...				620
Per cent. of sulphur eliminated..				80.6

In several localities the ore hearth has been operated on high-grade galena, containing about 80 per cent. lead, with an extraction of from 80 to 85 per cent. of the metal. In order to determine the adaptability of this grade of ore to the Newnam hearth a test run was made, the average results of which, per 8-hr. shift, are given below.

Note the very high extraction of metallic lead and the correspondingly low amounts of dust, fume, and slag made.

Test on Galena Concentrates, Containing 82.0 Per Cent. Lead and 11.2 Per Cent. Sulphur

	Pounds
Dry ore charged.....	14,436
Lead contents.....	11,837
Pig lead made.....	10,790
Gray slag made.....	1,075
Per cent. of coke breeze used.....	2.4
Per cent. of crushed limestone.....	2.0

Percentage of Total Lead in Products

Pig lead.....	91.15
Gray slag.....	4.25
Dust and fume.....	4.60
	100.00

Sulphur Elimination

	Pounds	Sulphur, Per Cent.	Sulphur, Pounds	Sulphur, Pounds
Dry ore charged.....	14,436	11.2		1,617
Gray slag.....	1,075	2.5	27	
Dust.....	254	11.1	28	
Burnt fume.....	508	5.9	30	85
Pounds of sulphur eliminated...				1,532
Per cent. of sulphur eliminated..				94.7

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Geology of the Ore Deposits of the Tintic Mining District

BY GUY W. CRANE,* B. A., EUREKA, UTAH

(New York Meeting, February, 1916)

I. INTRODUCTION

THE geology of the Tintic mining district, fully treated, would occupy an elaborate monograph. This less comprehensive paper is devoted primarily to the occurrence and origin of the orebodies of the district, while space is given to those phases of the general geology only which seem necessary to a proper setting of such a discussion.

The first geological report on the Tintic district to be issued was that by Tower and Smith, published in 1899, in the form of a paper in Part III of the *Nineteenth Annual Report of the U. S. Geological Survey*, and also as *Folio No. 65, Geologic Atlas of the United States*. About four years ago a second investigation was begun by Waldemar Lindgren and G. F. Loughlin, of the U. S. Geological Survey. With the exception of a paper on the oxidized zinc ores of the Tintic district by Mr. Loughlin, the results of their work are still unpublished. The writer's acquaintance with the district began in the fall of 1913, and has extended over a period of about two years.

1. Location

The Tintic district lies about 65 miles due south of Salt Lake City, and on the west central slope of the Tintic Mountains, a short range, which forms the connecting link between the Oquirrh Mountains on the north and the Canyon Range on the south—the group constituting the first of the Basin ranges to the west of the Wasatch Mountains.

The productive portion of the district embraces an area about 6 miles long and 2 miles wide, which is divided between Juab and Utah Counties. Within this area are the towns of Eureka, Mammoth, Robinson, Silver City, and Knightville. All these points are tapped by the Denver & Rio Grande and S. P., L. A. & S. L. Railways.

2. Topography

As in most of the Basin ranges, topographic relief is strong, dropping rapidly from peaks of 8,100 ft. above sea level to elevations of 5,600 ft. in Tintic Valley on the west and 4,500 ft. in Goshen Valley on the east.

*Mining Geologist.

The regularity of the range is broken by many lateral valleys, one of which has cut well back into the range, forming the low divide above the town of Eureka, through which the D. & R. G. Railway enters the district.

3. History

Tintic is among the oldest mining camps in Utah, being antedated only by Bingham, Rush Valley, and Little Cottonwood. Ore was first discovered in the monzonite about 1 mile east of Silver City in December, 1869, and the district was organized in the following spring. The first claim recorded was the Sunbeam, located Dec. 13, 1869, on the first discovery. The second location was made Jan. 3, 1870, on the Black Dragon, a little farther to the north. This was followed in quick succession by locations in the limestone, first on the Mammoth ledge near the middle of the district, and shortly after on the Eureka Hill ledge near its northern end.

Mining operations began at once with the construction of two mills and as many smelters during the first 12 months. Once located, the orebodies have been found to be fairly persistent, with new discoveries from month to month, and today there is good reason to believe that this record can be maintained for some time to come.

4. Production

During the early days of the district the production was about equally divided between the deposits in the igneous rocks at the south end of

*Total Metal and Gross Values Production of the Tintic Mining District to Jan. 1, 1914.**

	1869 to 1879, inclusive	1880 to 1903, inclusive	1904 to 1913, inclusive	Totals to Jan. 1, 1914
Gold: Ounces.....			831,114	
Value.....	\$475,451	\$11,096,747	\$17,180,602	\$28,752,800
Silver: Ounces.....	1,300,000	50,400,000	51,613,173	103,313,173
Value.....	\$1,616,130	\$38,620,870	\$27,343,438	\$67,580,438
Copper: Pounds.....	1,700,000	37,235,000	89,233,411	128,168,411
Value.....	\$478,852	\$6,081,148	\$13,473,378	\$20,033,378
Lead: Pounds.....	11,000,000	269,000,000	299,176,172	579,176,172
Value.....	\$631,146	\$12,073,854	\$13,869,125	\$26,574,125
Zinc: Pounds.....			7,306,281	7,306,281
Value.....			\$457,378	\$457,378
Total value.....	\$3,201,579	\$67,872,619	\$72,323,921	\$143,398,119

* Compiled from statistics of G. W. Tower and G. O. Smith, U. S. Geological Survey, V. C. Heikes in reports of 1904 to 1913, inclusive, and other sources.

the district and those in the limestone at the north end. But upon the exhaustion of the oxidized ores in the igneous rocks, due to high water-level, the balance of production was shifted to the deposits in the limestone; and since the closing of the Swansea mines at Silver City, in 1913, the entire production has been from the latter deposits.

II. AREAL GEOLOGY

The rocks of the Tintic district consist of Paleozoic sediments and a variety of igneous rocks of Tertiary age.

1. *Sedimentary Rocks*

These consist mainly of quartzite and limestone having a total thickness of 12,000 ft. or more. For economic and lithologic reasons they have been subdivided in descending section as follows:

Humbug sandstone	} Constituting the Humbug formation of the U. S. Geological Survey.
Humbug limestone	
Tetro limestone	} Constituting the Godiva limestone of the U. S. Geological Survey.
Carbonaceous shale	
Blue Fossiliferous limestone	
Chief Consolidated limestone	
Gemini limestone	
White-lime-shale	} Constituting the Eureka limestone of the U. S. Geological Survey.
Centennial limestone	
Golden Ray limestone	
Tintic slate	} Constituting the Tintic or Robinson quartzite of the U. S. Geological Survey.
Tintic quartzite	

At the top of the section is the Humbug sandstone, on the east slope of the Godiva Mountain, which is 224 ft. thick, consists mainly of calcareous sandstone with a few intercalated beds of arenaceous limestone, and is not known to be ore bearing.

The Humbug limestone, 378 ft. thick, consists mainly of nearly pure, coarse-grained, gray limestone with a few intercalated beds of arenaceous limestone of a yellowish buff color. This formation is well exposed on the upper east slope of Godiva Mountain and has carried most of the ore in the Godiva-Sioux Mountain and Iron Blossom ore zones.

The Tetro limestone, 355 ft. thick, consists with little variation of hard, fine-grained, blue, cherty limestone which is not known to be ore bearing.

The Carbonaceous shale, 160 ft. thick, is essentially a thinly bedded, black, shaly limestone, containing bands of black chert, and weathering to a platy or shelly surface. It is most unpromising as an ore-bearing medium and is nowhere known to be mineralized.

The Blue Fossiliferous limestone, 542 ft. thick, consists of 300 ft. of blue, cherty limestone in beds from 1 to 3 ft. thick, resting upon 242 ft.

of blue, highly fossiliferous limestone in thin beds separated by thin partings of impervious clay shale. This formation has been quite extensively prospected, and found to carry no orebodies of importance.

The Chief Consolidated limestone, 615 ft. thick, consists of a series of contrasted horizons which, in descending order, have the following characteristics:

80 ft. of very coarse-grained, massive, dolomitic limestone, on the whole quite favorable to mineralization.

64 ft. of fine-grained, siliceous, hard, brittle, black limestone, which is generally barren, though running the entire length of the ore zone in the Chief Consolidated mine.

38 ft. of medium-grained, fairly pure, blue limestone in thin beds, but with a few black shaly partings which appear to offer some resistance to mineralization.

31 ft. of coarse-grained, pure white to light-gray limestone which appears to be especially favorable to mineralization.

120 ft. of blue, flaky and fairly pure limestone which also appears to be especially favorable to mineralization.

72 ft. of coarse, mottled, light-gray, massive dolomitic limestone, readily susceptible to mineralization.

212 ft. of fine-grained, ashy gray, impure limestone with 14 or more intercalated thin beds of quartzitic sandstone. The horizon lies to the west of the ore zone proper, and its character is such as to discourage exploration for ore within its limits.

The Gemini limestone, 902 ft. thick, consists of 45 or more relatively thin alternating horizons of blue, gray, light-gray, and white limestone of varying texture and hardness, but all generally distinctly bedded. The formation is especially characterized by its purity, *i.e.*, lack of siliceous members, and general susceptibility to mineralization.

The White-lime-shale, 920 ft. thick, consists throughout of fine-grained, bluish gray, thinly bedded, shaly limestone with a few thin conglomeratic beds of the interformational type. It is generally extensively sheeted where folded, and weathers to a yellowish white, whence its name. It is characteristically unfavorable to mineralization, and is not known to be ore bearing.

The Centennial limestone, 798 ft. thick, consists of four relatively thick horizons of massive, dark-blue limestone interspaced by as many relatively thin horizons of thinly bedded, light-gray limestone and one 6-ft. member of thinly laminated, light-green shale. Except for the shale and two 20-ft. impure, sandy horizons, the entire formation is apparently favorable to mineralization.

The Golden Ray limestone, thickness 1,559 ft., consists of five thick horizons of heavily bedded, dark-blue to bluish gray, dolomitic limestone interspaced by five relatively thin horizons of fine-grained, light-

gray to white, thinly laminated, impure limestone. While this formation is not known to be ore bearing, a large part of it possesses characteristics which are apparently favorable to ore deposition in other ore-bearing horizons.

The Tintic slate, thickness 358 ft., consists essentially of thinly laminated, green slate with intercalated bands of impure, gray, banded limestone near the top and thin bands of brown, quartzitic slate near the bottom. The formation is highly sheeted due to folding. Its highly siliceous and wholly barren nature does not invite exploration.

The Tintic quartzite, the lowest member in the stratigraphic series, is not fully exposed, but has an estimated thickness of from 5,000 to 7,000 ft. It is for the most part a pure, compact, fine-grained quartzite, white to pink in color. Bedding indistinct and readily confused with a pronounced sheeting due to folding. Occasional interspaced thin beds of pebble-quartz-conglomerate break the general massive nature and lend a clue to the structure.

No special effort has been given to the determination of the age divisions in the above succession. The section affords, however, no evidence of an important stratigraphical break, such as a real erosion unconformity. The thin conglomerates in the White-lime-shale are of the interformational type, and represent only temporary disturbances of the normal state of deposition during a period of shallow-sea conditions. If the chronology as determined by Tower and Smith¹ is to stand, the succession certainly represents continuous deposition from Middle Cambrian well into Pennsylvanian time.

2. Igneous Rocks

Packard Rhyolite.—The igneous rocks consist of rhyolite, andesite, tuff and breccia, and monzonite, all of Tertiary age.

The Packard rhyolite, locally referred to as porphyry, occurs as surface flows and intrusive dikes. It occupies a large area in the northern end of the district and, underlying the town of Eureka, extends to the north and east in a range of hills of which Packard's Peak is the most prominent. Its thickness at this point probably exceeds 1,000 ft. The rhyolite varies in color from pink and light purple to light and dark gray. In texture it ranges from glassy to highly granular, with flow banding as a common characteristic.

No metalliferous deposits are known to occur with the Packard rhyolite. Its chief economic interest lies in the numerous springs issuing from the lower slopes, from which the entire water supply of Eureka and the mines in that section of the district is obtained.

¹ Tower and Smith: *Geology and Mining Industry of the Tintic District, Utah, 19th Annual Report, U. S. Geological Survey, Part III, p. 627.*

Andesite, Tuff, and Breccia.—Large areas of andesite with associated tuff and breccia lie to the east and south of the productive portion of the Tintic district. They are, for the most part, older than the metalliferous deposits and, having no genetic relation to the latter, they need not receive further attention here.

Sunbeam Monzonite.—The monzonite is confined to the southern half of the district, where it forms the country rock over an area 4 miles in length, north and south, and 2 miles wide. It is the youngest of the igneous rocks, and occurs in the form of an intrusive stock, cutting and enveloping fragments of all the earlier formations. On the north it lies in an irregular, but nearly vertical, contact with the limestones, which it has metamorphosed through a zone varying from 300 to 500 ft. in width. On the east and south it is in irregular contact with the andesite. Its western boundary is obscured by the overlapping, alluvium-filled valleys, Ruby Hollow and Diamond Gulch, which show it to have been deeply eroded.

In color it is light to dark gray, usually with greenish brown tinge. In texture it is normally evenly granular, but varies from porphyritic on its east and west boundaries to granitic where it is in vertical contact with the limestones.

What is known as the Swansea rhyolite is probably a rhyolitic phase of the monzonite and belongs to the same period of intrusion.

The monzonite is a formation of first importance, since it has been the mineralizing source of all the important orebodies in the Tintic district.

3. Structure

Folding.—The sedimentary formations are folded into a simple and fairly symmetrical overturned syncline, the major axis of which strikes north and south and pitches slightly to the north. The axial plane of the fold dips about 60° to the west, resulting in vertical and often overturned dips in the beds of the west limb of the syncline, while the angles of the dip of the beds in the east limb are correspondingly low, rarely exceeding 40°. By reason of its pitch, the syncline becomes wider to the north where the trough is folded into a series of crenulations, forming a wide synclinatorium. South of the Ajax fault, which extends from the Mammoth mine east to the Iron Blossom No. 3 mine, no synclinal structure is obtained. While the beds on the north of the fault are steeply inclined, conforming to the normal structure of the syncline, those on the south are relatively flat, lying with gentle dips to the east, in which direction no return dips are observable, because of a thick covering of volcanic rocks.

Faulting.—The folding of the sediments was accompanied by extensive faulting, fracturing, fissuring, and sheeting. The major faults are of the

transverse type, striking in a nearly east-west direction across the line of the major fold. These faults show lateral displacements ranging from 500 to 3,000 ft. The direction of throw is usually to the east on the south. Connecting the major faults at various angles are numerous minor faults showing displacements ranging up to 200 ft. Rotary faulting is common, as is shown by marked disparity in the attitude of the beds along the plane of the faults. The major faults are usually accompanied by one or more parallel faults, inclosing a breccia or sheared zone. The occasional occurrence of breccia zones and slickensides parallel to the bedding planes must be interpreted as evidence of some faulting of this type. None of the faults are known to extend into the igneous rocks, which are evidently of later origin. While it is probable that most of the faulting occurred during the period of folding, it is not unlikely that some faulting and fracturing accompanied the intrusion of the monzonite, though none of this origin has been recognized.

Fractures, Fissures, and Sheeting.—Fractures and fissures occur in both the igneous and sedimentary rocks. Sheeting is confined to the latter.

In the Sunbeam monzonite and its porphyritic phase, the Swansea rhyolite, are numerous ore-bearing fissures which have a general north-south direction, but vary from as much as N. 20° W. to N. 45° E. These fissures are of the stretch type, due to shrinkage accompanying the consolidation of the igneous intrusives. For this reason they do not extend into the adjacent sedimentary rocks.

The sediments during the period of their deformation were extensively fractured and sheeted, generally along shearing planes. The fractures, which are usually steeply inclined, are most numerous in the steeply dipping beds and in the vicinity of the major faults. They trend at all angles, those [in the northeast and southwest quadrants being the most abundant.

Sheeting has occurred in the quartzite, slate, and more impure shaly limestone beds, where these have been closely folded. It is best developed in the Tintic slate, and the White-lime-shale, along the western limb of the syncline. The Tintic quartzite also is extensively sheeted.

III. GEOLOGIC HISTORY

The geologic history of the district is, briefly, somewhat as follows:

During Paleozoic time the area was under water, as is evidenced by the 12,000 ft., more or less, of sediments of Paleozoic age. That conditions of sedimentation fluctuated considerably between shallow and deep waters, is confirmed by the alternation of thinly bedded shaly limestone, and shale with relatively pure massive limestones; but there is no evidence that, during this age, any portion of the area was elevated above sea level even for a brief interval.

Early in Mesozoic time extensive earth movements began, which resulted in the elevation and deformation of the Paleozoic sediments into virtually their present highly disturbed condition. Throughout this period of folding and faulting, erosive agents were active in carving the distorted strata into mountain ranges closely approaching the present forms. While folding had probably ceased some time before, the erosion continued right up to the Tertiary.

During Tertiary time, the greater portion of the area was buried beneath extensive flows of rhyolite and andesite, which were interspaced by short periods of erosion. Where the rhyolite flows overlapped the sediments, considerable quantities of iron oxide and ferruginous jasper were formed. These are described under the head of limestone-igneous contact deposits.

Following the extrusion of the Packard rhyolite and andesite, with at least a short period of erosion intervening, came the intrusion of the Sunbeam monzonite and its quartz-porphyry phase, the Swansea rhyolite.

This event brought about the second period of ore deposition, which resulted in the formation of the replacement deposits in limestone and was accompanied or soon followed by the deposition of the fissure-vein ores in the Sunbeam monzonite and Swansea rhyolite. Since the last of the volcanic eruptions and the period of ore deposition, there has been uninterrupted erosion, which has cut deeply into the accumulations of volcanic materials, exposing extensive areas of the sediments and, at a few points, the inclosed orebodies.

The products of this last period of erosion form the alluvium in the valleys and the talus on the upper slopes.

IV. THE ORE DEPOSITS

There are three types of ore deposits in the Tintic district: (1) limestone-igneous contact deposits; (2) deposits in igneous rocks; and (3) limestone replacement deposits.

1. *Limestone-Igneous Contact Deposits*

At numerous points along the contact of the limestone and the igneous rocks, particularly the Packard rhyolite, are thin deposits of iron and manganese oxide associated with more or less red and greenish yellow jasperoid and clay-ironstone. The jasperoid has been observed in contact with the limestone of which it is a replacement. The iron-manganese deposits occur both at the contact and filling cavities in the limestone near the contact.

Deposits of this type are widely distributed over the district, particularly its northern end, where, because of the resemblance of the

associated jasperoids to certain phases of the quartz in the productive limestone replacement deposits, they have received considerable attention from prospectors.

Locally small quantities of silver and gold have been reported; but, except for the iron ores which have been mined intermittently for fluxing purposes, these deposits have no economic importance.

2. Deposits in Igneous Rocks

The ore deposits in the igneous rocks are confined to the southern half of the district, where they occur as veins in well-defined stretch fissures in the Sunbeam monzonite and Swansea rhyolite. The veins are as a rule nearly vertical with frequent croppings at the surface. They have general north-south courses but vary from N. 20° W. to N. 45° E. One, the Sunbeam, has been worked through a length of 2,000 ft. on the strike, but most of them pinched out at a distance of a few hundred feet.

As shown by the old workings, the veins vary in width from a few inches to 10 ft., usually pinching with depth. The average width is about 2 ft. The ore in its unaltered state consisted of silver-lead sulphides and sulpharsenides, occurring in bands and lenses in a gangue of quartz and pyrite. These materials constituted about three-quarters of the mass of the vein. Considerable copper occurred in some of the southern mines, while those in the north have produced lead and silver almost exclusively. Gold occurred sparingly in but few of the deposits. Ground-water level was usually encountered at a depth of from 200 to 700 ft., depending upon the proximity of the sediments. Above water level, the ores assumed the usual oxidized character, with some secondary enrichment. Below ground-water level there was marked decrease in the value and size of the orebodies. This and water troubles usually caused an early cessation of mining, and today all these properties, with one or two exceptions, are indefinitely abandoned. Since the closing of the Swansea mines in 1913 there has been no noteworthy production from deposits of this type.

3. Replacements in Limestone

The orebodies of this type are confined to the northern half of the Tintic district, and include all the now productive mines. Mining operations have developed four nearly parallel ore-bearing zones, within which the orebodies appear to be virtually continuous, though at intervals the connections may be mere seams of quartz. These zones have a general north-south direction and may be defined as follows:

1. The Eureka zone, including the Centennial Eureka mine on the south and extending through the Eureka Hill, Bullion Beck, and Gemini mines to the Ridge and Valley mine on the north.

2. The Mammoth zone, beginning at the Black Jack mine on the south and extending through the Lower Mammoth, Phoenix, Gold Chain, Mammoth, Grand Central, Victoria, Eagle, and Blue Bell mines to the Chief Consolidated mine on the north.

3. The Godiva-Sioux Mountain zone, beginning at the North Star mine on the south and extending through the Red Rose, Carisa, Northern Spy, Utah, Uncle Sam, Yankee, May Day, and Apex mines to the Godiva mine on the north.

4. The Iron Blossom zone, beginning at the Dragon Consolidated mine on the south and extending through the Black Dragon, Governor, Iron Blossom Nos. 1 and 3, Sioux Consolidated, Colorado Nos. 1 and 2, and Beck Tunnel No. 1 mines to the Beck Tunnel No. 2 mine on the north.

As might be inferred, these four zones include all the mines that have produced this type of ore. The Eureka zone is the shortest, and at its northern end is divided into two or more channels. At its southern end, it appears to connect with the Mammoth zone, through the Grand Central mine, in which case it is a branch of the Mammoth zone. These two zones lie on the west limb of the syncline, where the beds are nearly vertical and their orebodies have great vertical range.

The other zones are well defined throughout their length of about 2 miles and apparently have no interconnections. They lie near the trough of the syncline and their orebodies are correspondingly shallow.

Occurrence of the Ore.—Like replacement deposits in general, those of Tintic are very irregular in form, but in both shape and size they are governed within certain limits by the associated structures. This explains the dissimilarity in occurrence of the orebodies in the different zones.

In the Eureka and Mammoth zones, which are in the nearly vertical beds of the west limb of the syncline, ore has been mined from the surface to the depth of 1,800 ft.; the orebodies as a rule conforming very closely to the bedding in both strike and dip.

If any one form of orebody can be said to be typical of these zones, it is that of an irregular sheet, pinching and swelling along the line of the beds, and feathering upward into chimneys or pipes.

In an earlier report on the Tintic district² reference is frequently made to the north-south fissures, which, it is contended, have governed the course of the ore channels. The writer's investigations have not confirmed the common presence of such north-south fissures within the ore zones, but, on the contrary, have gone to show that practically all the major faulting and fissuring in the district is of the nearly east-west or transverse type.

² Tower and Smith: *Geology and Mining Industry of the Tintic District, Utah. 19th Annual Report, U. S. Geological Survey, Part III.*

Where the ore channel is crossed by faults or zones of fracture, there is usually an enlargement of the orebody along the line of intersection of the faulting and the bedding. This gives rise to another very common type of orebody, characteristic of the western zones, where it is usually the most largely productive form.

At the intersection of the ore channel by a fault, it is not an uncommon occurrence to find that the line of mineralization turns from the plane of the beds and follows the fault for distances up to 100 ft. or more, before again taking the usual course along the beds. Occurrences of this kind probably also account for the occasional division of a single ore channel into two or more parallel ore channels.

In the Godiva-Sioux Mountain zone, the known orebodies have smaller vertical extent but greater lateral variation than do those in either the Eureka or Mammoth zones. This is probably due to the fact that the Godiva-Sioux Mountain zone lies nearer the trough of the syncline where the beds dip at relatively lower angles, seldom exceeding 60° and sometimes as low as 15° .

In the Iron Blossom zone, particularly in its northern half, we find still greater restriction in both vertical and lateral extent of the ore than in any of the other ore zones. This, again, can be attributed to the controlling influence of the structure, since this portion of the zone takes a course directly in the trough of the syncline. The south half of the zone traverses beds dipping at angles up to 30° , and here the orebodies resemble more closely in form those of the Godiva-Sioux Mountain zone.

Influence of Texture and Composition.—Next in importance to structural control is that of the texture and composition of the limestone. Other things being equal, the coarser-grained, softer, and less siliceous limestones are more susceptible to mineralization than are the fine-grained, more siliceous and harder ones, while the associated shales, sandstones, slates, and quartzite are uniformly barren. This principle is well exemplified in all the Tintic replacement deposits, as well as in the replacement deposits of other districts.

Among the several formations described above, there are only four which are known to be ore bearing. These are the Centennial, Gemini, Chief Consolidated, and Humbug limestones. In each case the ore-bearing formation contains several horizons of coarse, soft, and relatively pure limestone, while the remaining formations are more generally fine-grained, hard and siliceous, in that they carry varying proportions of shale and sand.

The influence of magnesia in the limestone is not clearly shown. One dolomitic horizon in the Chief Consolidated formation is quite generally mineralized, while another is only sparingly so. The dolomites are usually coarse grained—a favorable factor, which may offset the disadvantages of difference of solubility.

Zone of Oxidation.—By reason of the folded and broken state of the limestone beds, the ground-water level is extremely low (about 4,765 ft. above sea level), in the sedimentary rocks. This puts it at depths beneath the surface of from 2,300 ft. in the higher properties to 1,500 ft. in those having shafts at relatively lower positions. This condition has led to more or less complete oxidation of the original sulphide ores, with some downward segregation of the ore minerals, and some enrichment due to shrinkage in volume.

V. THE ORE

Very little ore has been mined below ground-water level; but developments have demonstrated its continuity into that zone, and something as to its original sulphide character. Above the water table the ore is largely oxidized.

1. *Physical Character*

The oxidized ore is usually massive and granular and quite without structure. A distinct banding, observed locally, is attributed by some to the preservation of an original banding in the replaced limestone. Some soft ocherous material, which usually caps the orebodies, is thinly laminated or stratified—the effect of transportation and redeposition during the process of oxidation. Material of this type is usually found capping the larger bodies of oxidized ore and is encountered more particularly in the higher levels of the oxidized zone.

2. *Mineral Composition*

The ore minerals contain gold, silver, lead, copper, and zinc, in a gangue composed mainly of quartz and some pyrite in the unaltered ore, but with important quantities of calcite and iron and manganese oxides, and smaller amounts of barite and dolomite in the oxidized ore.

Gold is seldom visible, but occurs locally in the native state. Silver occurs native, and where unoxidized as the sulphide (argentite) and the antimonial sulphide (stephanite); but the original compounds are usually altered to the chloride (cerargyrite, or horn silver). Lead occurs originally as the sulphide (galena), but is largely altered to the carbonate (cerussite), with smaller amounts of the sulphate (anglesite), and one or two rarer forms. Copper occurs originally as the sulpharsenide (enargite), but is usually altered to the hydrous silicate (chrysocolla) and the hydrous carbonates (malachite and azurite). Some native copper has been observed which may be in part original. Zinc occurs originally as the sulphide (sphalerite), but is usually altered to the carbonate (smithsonite) or the hydrous silicate (calamine), with small amounts of one or two rarer forms.

3. *Distribution of the Ore Minerals*

From the earliest days an important variation in the north and south distribution of the ore minerals in the several ore zones has been noted. It appears that gold-copper ores predominate at the south end of the zones, gradually giving place to increasing proportions of silver-lead ores to the north, where the latter minerals predominate. As a result, the mines in the vicinity of Mammoth have been chiefly producers of copper and gold, while those in the vicinity of Eureka are chiefly producers of silver and lead. Zinc is found associated with the ore of both ends of the district, probably more frequently in the mines at the north. Vertically the ore zones show no such striking variation in mineral content; but generally the good gold stopes are in the upper levels.

The vertical extent of an ore zone is governed primarily by the accompanying structures. The more nearly vertical are the inclosing beds the greater the vertical range of the orebody. The presence of faults and fissures also favors greater vertical range, irrespective of the angle of the bedding. Their influence is greatest, of course, where the bedding is nearly vertical. The large orebody in the south end of the Chief Consolidated mine, which has a vertical range of 1,000 ft. or more, rakes along the inclined line of intersection of vertical beds and a large cross break.

4. *Origin of the Ore*

The occurrence of the Tintic deposits both within the monzonite and in the immediately adjacent limestones; the character of the mineralization, which is now generally considered to be due to the action of hot solutions; the progressive change in the mineral composition of the ore with increased distance from the monzonite—all point to this rock as the prime factor in the mineralization. In this respect, the deposits of Tintic are like the many others in the southwestern part of the United States, and particularly in western Utah, which are as a rule closely connected in origin with igneous (commonly monzonite) intrusions.

The localization of the deposits within the monzonite was determined by the formation of stretch fissures within the congealing magma, which furnished channels of circulation for the hot solutions that were given off during the period of cooling. These hot solutions chloritized the ferromagnesian minerals, sericitized the feldspars of the wall rocks, and filled the fissures with an aggregate of quartz, pyrite, galena, sphalerite, barite, and some copper, silver, and gold minerals, probably because of a decrease in pressure and temperature as the solutions approached the surface. Alteration above ground-water level has changed these minerals to their oxidized equivalents, with some secondary enrichment.

The localization of the deposits in the limestone was controlled in

part by the attitude and character of the beds and in part by the presence of fissures, some of which were probably produced during the period of monzonite intrusion.

Following the fissures caused by the intrusion, the hot mineral-bearing solutions found their way through the fractured and metamorphosed zone into the unaltered limestone, where they naturally took the easiest avenue afforded. This, in the absence of open faults and fissures, was the plane of the bedding.

Where the dip of the beds approached the vertical, the solutions found channels at various elevations, ascending or descending as conditions allowed, but laterally confined to relatively narrow limits. Flat-lying and low-pitching beds, on the other hand, have tended to confine the solutions to a more nearly horizontal course, at the same time permitting greater lateral shift.

At the intersection of faults, the solutions were allowed to spread out both vertically and laterally, producing orebodies of large size and great vertical range.

The process of mineralization has been primarily a replacement of the limestone with quartz and varying proportions of copper, lead, zinc, and silver sulphides, presumably due to decreasing temperature and pressure in the solutions as they run their northerly course.

Soft, coarse-grained, nearly pure limestones have been much more readily replaced by the solutions than the harder, fine grained, and more siliceous beds. Certain horizons of the latter type, even where directly in the course of the mineralized zone, are found to be uniformly barren.

VI. PROSPECTS FOR THE FUTURE

Of the four zones so far developed, not one has been traced to a logically demonstrated terminus; and it is by no means established that similar and equally productive zones do not exist, still undiscovered, in the district.

Since the inception of mining, following the first discovery in 1869, the trend of production in the district has been steadily upward. Great mines have come and gone; but there have been always new ones to take their places. The greatest production in the district's history was attained but two years ago, and that of the coming twelve months promises to exceed it. In view of these facts, and of the tendency to better and more improved methods of mining as our knowledge of the deposits is extended, the conclusion is inevitable that the Tintic district has still to experience the most prosperous period in its history.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close April 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Measurement of the Temperature Drop in Blast-Furnace Hot-Blast Mains

BY R. J. WYSOR,* SOUTH BETHLEHEM, PA.

(New York Meeting, February, 1916)

MORE than two years ago, in making efficiency tests on our hot-blast stoves, I was surprised to discover a marked difference in temperature as indicated by a pyrometer inserted near a stove on blast, and the regular recording instrument situated near the bustle pipe. A special hole was then provided at the junction of the hot-blast main and the bustle pipe, and portable pyrometers with bare fire ends were used, both at this point and in the stove necks. I had occasion to make a number of such tests on various stoves on the same furnace, the difference in temperature as shown by the two pyrometers varying chiefly with the temperature of the blast, and the distance of the stove from the furnace. However, this difference was always great, often averaging above 100° F., for the entire period of a stove on blast. Later, experiments were made with similar results on another group of stoves. The pyrometers used were properly standardized and checked against each other before and after each test.

However, there was still a slight doubt as to the accuracy of the results obtained, as subsequently explained. But I have obtained corroboration of the magnitude of heat losses from blast-furnace hot-blast mains from two independent sources. I have learned from H. A. Brassert, Superintendent of Blast Furnaces, South Works, Illinois Steel Co., that he has recently done some experimental work similar to my own, and found temperature drops of the same order of magnitude. Also, C. R. Lyle, of the Armstrong Cork Co., has made some interesting theoretical calculations, based on the thermal conductivity of fire brick, for the particular conditions of several of our tests; and his theoretical interpolations, all facts being considered, check our actual observations fairly well. I understand that Mr. Lyle expects to publish the results of his investigations.

METHOD OF CONDUCTING THE TESTS

The location of the pyrometer fire ends is shown in Fig. 1, a plan of Bethlehem Steel Co.'s "G" blast furnace, hot-blast stoves, and main. Bare fire ends were used at both positions "A" and "B," being inserted

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so as to project about 8 in. inside the fire-brick lining of the horizontal stove neck and hot-blast main respectively. Readings were taken simultaneously at both positions at 3 to 5 min. intervals from the time the stove was put on until it was taken off blast. As a check on pyrometer B, we had the regular recording pyrometer located only a few feet away in the hot-blast main. The results of the latter were slightly

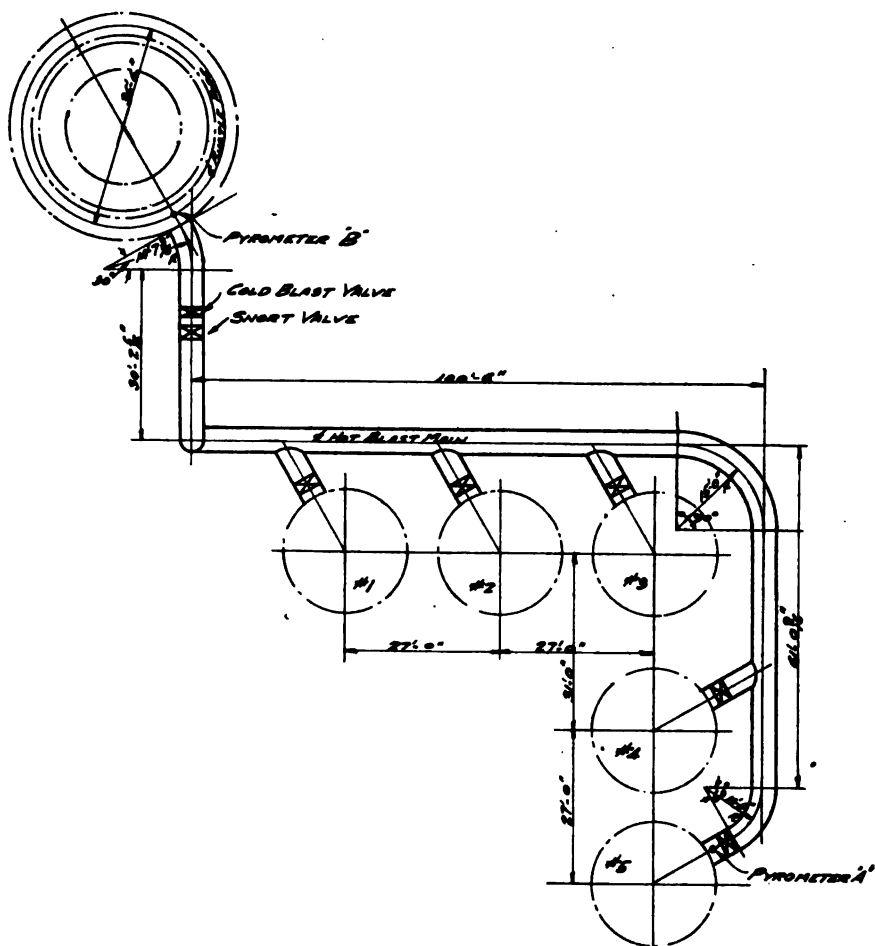


FIG. 1.—PLAN SHOWING HOT-BLAST MAIN AND CONNECTIONS—"G" FURNACE—AND LOCATION OF PYROMETERS.

lower on account of the fire end being shrouded with asbestos packing in a metal tube.

Three possible sources of error were suspected: high readings of pyrometer A, due to radiation from heated brick work of the stove neck or adjacent parts of the stove; uneven temperature of the hot blast passing through the cross-section in which pyrometer A was located;

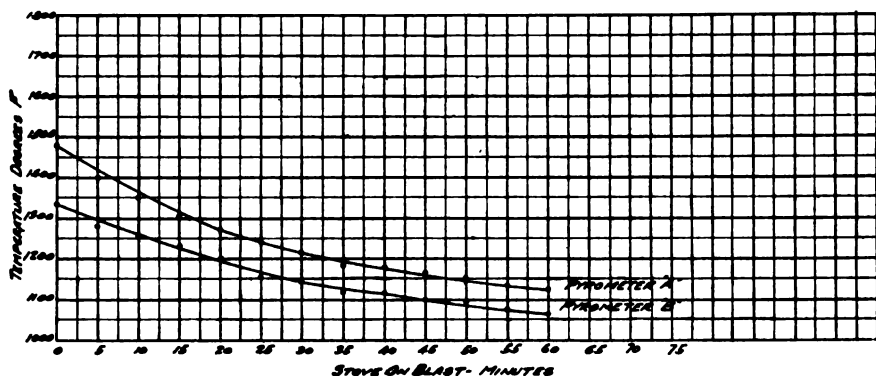


FIG. 2.—TEST 1. CURVES SHOWING DIFFERENCE IN TEMPERATURE BETWEEN POINTS IN HOT-BLAST MAIN, "E" FURNACE, WHILE STOVE WAS ON BLAST, OCT. 10, 1912.
Pyrometer A, between stove No. 5 and hot-blast valve.
Pyrometer B, at junction of main and bustle pipe.

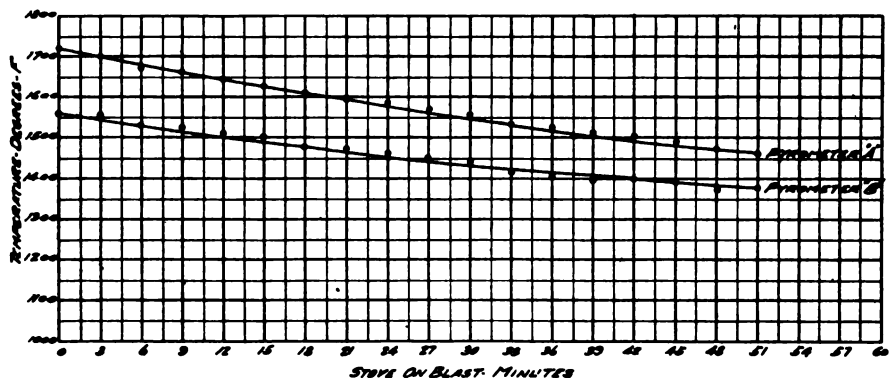


FIG. 3.—TEST 2. CURVES SHOWING DIFFERENCE IN TEMPERATURE BETWEEN POINTS IN HOT-BLAST MAIN, "G" FURNACE, WHILE STOVE WAS ON BLAST, NOV. 6, 1913.
Pyrometer A, between stove No. 5 and hot-blast valve.
Pyrometer B, at junction of main and bustle pipe.

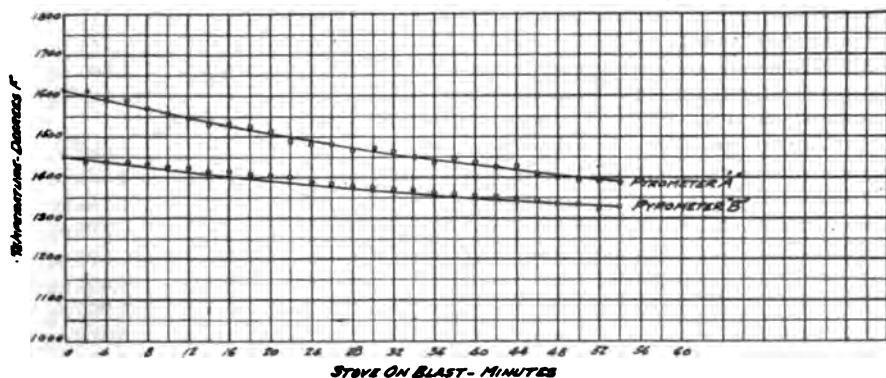


FIG. 4.—TEST 3. CURVES SHOWING DIFFERENCE IN TEMPERATURE BETWEEN POINTS IN HOT-BLAST MAIN, "G" FURNACE, WHILE STOVE WAS ON BLAST, NOV. 10, 1913.
Pyrometer A, between stove No. 5 and hot-blast valve.
Pyrometer B, at junction of main and bustle pipe.

low readings of pyrometer B due to leakage of the cold-blast valve. Both direct and indirect information leads me to believe it unlikely that appreciable error has occurred due to any of these causes.

The accompanying diagrams and tables, selected from a number, show typical conditions for the two furnaces in question. Tests were not made on other furnaces, because of the objectionable feature of breaking out brick work in the hot-blast main, with resulting danger of hot spots.

Unfortunately, no tests were made on comparatively low blast heats. On account of smelting hard, lumpy ores, we usually carry high blast temperatures at Bethlehem, and generally operate five stoves. The two runs on No. 5 stove, "G" furnace, show unusually high blast temperatures and this particular hot-blast main, for engineering reasons, is considerably longer than normal. The results show the high losses obtaining for such conditions. The "E" furnace test is more nearly representative, though the average temperature is still higher than for furnaces smelting large percentages of Mesabi ores.

General Data and Dimensions

For purposes of calculation the following data in connection with "G" furnace and hot-blast main, with connections, will be of interest. The furnace is rated at 500 tons capacity. It is served by five three-pass McClure stoves, 22 ft. diameter by 100 ft. high, connecting to the hot-blast main through Kennedy-Morrison hot-blast valves. The diameter, and the thickness of brick work in all the 500-ton furnace hot-blast mains are the same.

Outside diameter of hot-blast main.....	48¾ in.
Thickness of steel shell.....	¾ in.
Thickness of fire-brick lining.....	9 in.
Length of hot-blast main on center line from hot-blast valve, No. 5 stove, to bustle pipe.....	192 ft.
Length of bustle pipe on center line.....	115 ft.
Distance between pyrometers A and B.....	218 ft.
Approximate total radiating surface corresponding to this distance....	2,606 sq. ft.
Approximate total radiating surface of five stoves.....	38,500 sq. ft.
Approximate total radiating surface of five hot-blast valves and connections to stoves and main.....	1,286 sq. ft.
Approximate total radiating surface of hot-blast main proper.....	2,373 sq. ft.
Approximate total radiating surface of bustle pipe.....	1,474 sq. ft.
Approximate total radiating surface of 10 leg pipes.....	370 sq. ft.
Approximate total radiating surface of 10 blow pipes and tuyères....	110 sq. ft.
Approximate total radiating surface between No. 5 stove and furnace..	4,560 sq. ft.
Approximate total radiating surface between No. 1 stove and furnace..	2,965 sq. ft.
Approximate total radiating surface between all stoves and furnaces..	5,613 sq. ft.

For most of its length, the hot-blast main is supported 15 ft. above the ground. The stoves and entire hot-blast system are protected with black stack paint.

LOGS OF THREE TYPICAL TESTS

The figures given under each of the subsequent tests represent actual conditions at the furnace at the time observed, except the wind velocities, which are averages for the day obtained by the Philadelphia station of the Weather Bureau; little significance is to be attached to them. The hot-blast mains of both of the furnaces concerned are fairly well protected from prevailing winds, and the actual velocity of the air passing over them probably did not exceed 4 miles per hour in any test.

Of course, the exact condition of the brick work in the hot-blast main could not be determined at the time of the tests. Only slight repairs were found to be necessary when the mains were opened after the preceding runs on the two furnaces concerned; and after being blown out again, the mains were still in fair condition, and no general repairs were necessary. Obviously we may assume that no great erosion of the brick throughout the hot-blast system had occurred up to the time of the test.

No pyrometer readings were taken for the first few minutes after the stove was changed from gas to blast.

Test No. 1

Date.....	Oct. 10, 1912
Furnace.....	"E"
Product.....	Standard basic
Time furnace was in blast prior to test.....	7 months
Time furnace was in blast after test.....	19 months
Stove.....	No. 5
Average atmospheric temperature.....	72° F.
Average wind velocity, miles per hour.....	5.8
Precipitation.....	None
Average blast pressure.....	16.3 lb.
Average free air blown per minute, at 62° F. and 30 in. barometer..	32,000 cu. ft.

Pyrometer Readings

Time A. M.	Pyrometer A Degrees F.	Pyrometer B Degrees F.
10.20	1,480	1,330
10.25	1,400	1,280
10.30	1,350	1,255
10.35	1,310	1,230
10.40	1,275	1,200
10.45	1,235	1,160
10.50	1,215	1,145
10.55	1,185	1,120
11.00	1,180	1,110
11.05	1,165	1,100
11.10	1,150	1,095
11.15	1,130	1,075
11.20	1,120	1,060
Average	1,246	1,166

Total time stove was on blast..... 65 min.

Direct average difference in temperature between pyrometers A and B. 79° F.

Test No. 2

Date.....	Nov. 6, 1913
Furnace.....	"G"
Product.....	Bessemer
Time furnace was in blast before test.....	13 months
Time furnace was in blast after test.....	13 months
Stove.....	No. 5
Average atmospheric temperature.....	60° F.
Average wind velocity, miles per hour.....	6.1
Precipitation.....	None
Average blast pressure.....	19.0 lb.
Average free air blown per minute, at 62° F. and 30 in. barometer.....	33,500 cu. ft.

Pyrometer Readings

Time P. M.	Pyrometer A Degrees F.	Pyrometer B Degrees F.
3.00	1,720	1,560
3.03	1,700	1,550
3.06	1,670	1,530
3.09	1,660	1,520
3.12	1,645	1,510
3.15	1,625	1,500
3.18	1,610	1,480
3.21	1,595	1,470
3.24	1,585	1,460
3.27	1,570	1,450
3.30	1,550	1,440
3.33	1,535	1,420
3.36	1,525	1,410
3.39	1,510	1,400
3.42	1,500	1,400
3.45	1,490	1,390
3.48	1,475	1,380
3.51	1,460	1,380
Average	1,579	1,459

Total time stove was on blast..... 55 min.

Direct average difference in temperature between pyrometers A and B. 120° F.

Test No. 3

Date.....	Nov. 10, 1913
Furnace.....	"G"
Product.....	Bessemer
Time furnace was in blast before test.....	13 months
Time furnace was in blast after test.....	13 months
Stove.....	No. 5
Average atmospheric temperature.....	51° F.
Average wind velocity, miles per hour.....	15.7
Precipitation.....	None
Average blast pressure.....	18.6 lb.
Average free air blown per minute, at 62° F. and 30 in. barometer...	33,500 cu. ft.

Pyrometer Readings

Time P. M.	Pyrometer A Degrees F.	Pyrometer B Degrees F.
3.20	1,610	1,450
3.22	1,610	1,440
3.24	1,590	1,440
3.26	1,590	1,435
3.28	1,565	1,430
3.30	1,555	1,425
3.32	1,545	1,420
3.34	1,530	1,410
3.36	1,530	1,410
3.38	1,520	1,405
3.40	1,510	1,400
3.42	1,490	1,395
3.44	1,485	1,385
3.46	1,480	1,380
3.48	1,470	1,375
3.50	1,470	1,370
3.52	1,460	1,365
3.54	1,450	1,365
3.56	1,440	1,360
3.58	1,440	1,355
4.00	1,435	1,350
4.02	1,425	1,350
4.04	1,420	1,345
4.06	1,400	1,340
4.08	1,400	1,335
4.10	1,395	1,330
4.12	1,395	1,325
4.14	1,390	1,320
Average	1,486	1,382

Total time stove was on blast..... 54 min.
 Direct average difference in temperature..... 104 °F.

DISCUSSION OF TEST DATA

For sake of illustration, the approximate amount of heat lost between the farthest stove and the bustle pipe, according to Test No. 3 will be calculated.

Cubic feet free air blown per minute at 62° F. and 30 in. barometer. 33,500
 Pounds free air blown per minute at 62° F. and 30 in. barometer.... 2,548
 Average temperature at stove No. 5..... 1,486° F.
 Average temperature at bustle pipe..... 1,382° F.
 Average difference in temperature..... 104° F.

Using specific-heat data on the basis of constant pressure, we find:

Total sensible heat (above 62° F.) in air blown per minute at stove No. 5, approximately.....	926,400 B.t.u.
Total sensible heat (above 62° F.) in air blown per minute at bustle pipe, approximately.....	856,400 B.t.u.
Loss in sensible heat per minute.....	70,000 B.t.u.
Average loss per square foot surface per minute.....	26.86 per cent.
Percentage loss in sensible heat above 62° F.....	7.7 per cent.

If there is an appreciable pressure drop in the hot-blast main due to friction, and to heat lost by radiation and conduction, part of the above temperature and sensible-heat loss would, of course, be accounted for as due to expansion of the blast with concomitant absorption of heat. We have made no tests to ascertain the actual change of pressure between the stoves and the furnace, nor have I seen any data published on this subject. However, whatever may be the loss of temperature due directly to blast expansion, it is reasonable to suppose that expansion of the blast in turn is largely caused by heat lost by radiation and conduction into the atmosphere.

From the foregoing data it will be seen that in the tests on No. 5 stove, "G" furnace, we have measured the temperature drop due to only about three-fifths of the total radiating and conducting surface between the last stove and the furnace, along the direct path of the blast. It will be realized, of course, that there are also considerable losses between the main and the hot-blast valves of the other stoves. While the temperature will not continue to drop in the brick-lined bustle pipe and leg pipes as rapidly as in the outlying sections of hot-blast main, it will still be appreciable. And though the total radiating and conducting surface of the bare blow pipes and water-cooled tuyères is relatively small, the loss of heat per unit area is relatively great.

In the course of this investigation, I determined the average thermal loss due to cooling water from each hot-blast valve, with an average blast temperature of 1,360° F. and volume of about 33,000 cu. ft. free air per minute under standard conditions, to be approximately 10,000 B.t.u. per minute. This was done simply by measuring the discharge water from the valve and from the valve seat, and determining the temperature of affluent and effluent water. Under the conditions of the above test No. 3, this would correspond to at least 15 per cent. of the total heat loss measured in the part of the hot-blast system under investigation. The large water-cooled surfaces of the mushroom type of valve in general use, and the high thermal conductivity of these bronze or copper parts, accounts for the considerable heat loss. In this connection it is interesting to note that Mr. Brassert is installing a new type of gate valve, having much smaller water-cooled surface in contact with the blast.

The greatest heat loss in the hot-blast system occurs, of course, when the stove farthest from the furnace is on blast. However, when one of the nearer stoves is on blast, there must still be considerable radiation and conduction losses from the idle portions of the main and connections.

It is not my purpose to discuss at length in this paper the radiation and conduction losses from hot-blast stoves. A good deal has been published on this subject and most blast-furnace superintendents know about what losses are chargeable to this source.

We are now experimenting at Bethlehem with a special insulating brick for reducing heat losses from the hot-blast main. Two straight sections of the main, about 15 ft. long each, between hot-blast valve connections, have been lined with $4\frac{1}{2}$ in. of special insulating brick, back of $4\frac{1}{2}$ in. of fire brick, and have been in service several months. Although an appreciably lower temperature of the shell on the outside of these sections was found than on other sections of the same main, the difference was not as great as had been calculated, doubtless by reason of the high thermal conductivity of the steel shell, with consequent rapid transmission between adjacent sections. It was not deemed wise to use these bricks in bends of the main, or in the short connections to hot-blast valves or stoves. Where there is a change of direction, fire brick are eroded more rapidly by the "sand-blast" action of dust in the hot blast; and if the inner course of fire brick should be pierced, the soft insulating brick would be cut out with comparative rapidity. This doubtless would be likewise the fate of any other ordinary insulating material similarly used.

Estimates have also been prepared for external insulation with $1\frac{1}{2}$ in. pipe-covering blocks, a 1-in. air space being provided. Besides the possible objections to appearance, first cost, and maintenance, the question as to possible overheating of the shell, with consequent warping or unequal expansion, must be considered.

In the construction of a new hot-blast main, the logical plan would apparently be to design it of sufficient diameter to permit lining with 9 in. of fire brick and a backing of several inches of insulating material.

CONCLUSIONS

1. There is, between the stoves and the blast furnace, a very marked drop in the temperature of the hot blast, which is not only serious as to temperature but signifies also a thermal loss. It is affected by many variables, such as size and length of hot-blast mains and connections, thickness of fire-brick linings, volume and initial temperature of air passing per unit of time, temperature and wind velocity of the outside air, extent of water cooling, etc. It is probable that, at a great many furnace plants in the country, these losses are of sufficient magnitude to

justify an attempt at their reduction. The present tendency is toward higher blast heats on furnaces carrying large percentages of Mesabi ores in their mixtures; and high heats have always been desirable in smelting hard, coarse ores, such as the magnetites so generally used in the East. In the manufacture of spiegel and ferro-manganese, the maintenance of high blast temperatures is of particular importance. In all plants where economy in the use of blast-furnace gas must be practiced, on account of its large consumption in gas engines or under boilers, this is likewise a matter of special interest.

2. That insufficient insulation in the hot-blast main is the cause of serious temperature and heat losses, will be readily recognized. It seems logical and practicable to prevent a large percentage of these losses by means of one of the methods of insulation suggested in this paper. With such insulation, other operating conditions being normal, we would expect higher available blast heats with resulting fuel economy, or a smaller quantity of fuel gas necessary for heating the stoves. I believe that blast-furnace managers and superintendents generally will be interested in a study of better insulation of the hot-blast main and stoves.

In conclusion, I wish to acknowledge the practical co-operation of one of my assistants, H. A. Floyd, in securing the data used in this paper.

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The Iron Deposits of Daiquiri, Cuba

BY WALDEMAR LINDGREN AND CLYDE P. ROSS, BOSTON, MASS.

(New York Meeting, February, 1916)

Introduction

To the miner, as well as to the geologist, the eastern part of Cuba is a most interesting region. Here we find, in contrast to the moderate relief predominating elsewhere in the island, an imposing mountain range, the Sierra Maestra, extending east and west parallel to the coast, its precipitous front facing the blue Caribbean Sea. In a geological sense this range is largely an unknown land, the only well-explored region being that in the vicinity of Santiago.

The range also contains the most important mineral deposits of the island. They comprise, first, a series of iron deposits yielding a partly hematitized magnetite with low content of phosphorus; second, a remarkable copper-bearing vein at El Cobre; and third, various manganese deposits. Probably more deposits will be found, for the larger and western part of the range is as yet little explored and its slopes are covered by a thick tropical jungle.

The following notes are based on a short visit by the senior author in January and February of 1914 to the mines at Daiquiri, El Cobre, and Mayari. For many courtesies and great assistance he is deeply obliged to Charles F. Rand, President of the Spanish-American Iron Co.; to George W. Pfeiffer, General Manager, and to the several members of his staff.

Geological Features of the Sierra Maestra

For our knowledge of the geology of the Sierra Maestra we are indebted to reconnaissances by Fernandez de Castro, R. T. Hill, and to W. C. Hayes, T. W. Vaughan, and A. C. Spencer. The summary by the latter three geologists furnishes the best guide to the region.¹ The range is undoubtedly outlined by a great east-west dislocation. Its southern slope drops abruptly to the sea, while on the north side, a

¹ C. W. Hayes, T. W. Vaughan, A. C. Spencer: *Report on a Geological Reconnaissance of Cuba*, pp. 69 to 83 (Washington, 1901).

much gentler declivity leads down to the rolling plateau of Tertiary limestone which occupies much of the adjacent part of the island. Viewed from the sea near Santiago or Daiquiri the summit of the range appears as an undulating line with an elevation of from 2,000 to 3,000 ft. and about 12 km from the sea. From this crest line an abrupt slope leads down to jungle lands little above the level of the sea, and separated from it by low coast hills, covered by a veneer of coral rock.

The range back of Santiago and Daiquiri seems to end before Guantanamo Bay is reached. At this place the coral limestones of the coast appear to reach far inland, but east of the bay and from here to Cape Maysi the mountains, here dry-looking and covered by brush, rise abruptly from the sea to a level skyline at an elevation of about 2,000 ft., the summit evidently forming a plateau, which probably extends northward, connecting with the high plateaus of Baracoa and Mayari.

Back of Santiago the railroad pass at Cristo is only 560 ft. above the sea while the road crossing the range at Boniato Hill reaches the summit 1,000 ft. above the tide and 12 km. from Santiago. West of Santiago there is a steep and high coast range separated by the Cobre Valley from the lower westward continuation of what may be called the Sierra Boniato. Farther west this coast range becomes much higher, culminating at Tarquino Peak (elevation 8,400 ft.) and dropping off precipitously into the sea.

The geological elements which make up the range near Santiago are as follows:

1. Recent coral rock veneering the coast hills.
2. Oligocene limestones and marls underlying the coral rocks along the low land of the coast.
3. A very thick series of greenstone tuffs and agglomerates dipping northward and well exposed in the Cobre Valley. Age unknown.
4. A thick series of heavy-bedded gray limestone well exposed on the road across the Sierra Boniato north of Santiago. These beds dip north steeply at the base of the ridge, more gently at the summit. Age unknown, but unless faulted they should overlie the tuffs of El Cobre.
5. Andesites, rhyolites, greenstones, and tuffs near Cristo and near the top of the high range back of Daiquiri; probably later than the limestones.
6. A large area of intrusive diorite or diorite porphyry with inclosed fragments of an older, bedded limestone. This diorite is said to form the seaward part of Sierra Cobre and occupies the foot hills and the basal part of the high ridge back of Daiquiri. The iron deposits are found in this diorite. The age of the included limestone and of the diorite is unknown. The idea of the geologists who have done the most work in this section seems to be that the lavas and tuffs and associated limestones are of Eocene age and that they rest on a basement of diorite.

Consequently the limestone included in the diorite must be older, possibly Paleozoic. While not able to contradict this statement, we believe that there is a possibility that the included limestones may be of the same age as those of Sierra Boniato and that consequently the great diorite intrusion might have formed the last feature of the volcanic and the sedimentary period represented by the Cobre and Boniato rocks.

Iron Deposits in the Diorite

As stated above, dioritic rocks occupy a wide area along the coast from the Sierra Cobre to Guantanamo Bay. In the diorite and close to the coast line are found a considerable number of local and limited masses of iron ore which are known to extend from a point 36 miles west of Santiago to Sigua, 25 miles east of that place. There are several mines along this belt of iron deposits; El Cuero mine lies west of Santiago. The Juragua properties owned by the Pennsylvania Steel Co. are at Sevilla and Firmeza, from 5 to 10 miles east of the city. The Daiquiri properties owned by the Spanish-American Iron Co. include the Vinent and the Berraco groups, the former comprising the Lola Hill mines described in this paper. The Sigua deposits lie near the coast and still farther east.

The Juragua mines were the earliest opened in this belt and production began in 1884, while the mines at Daiquiri started shipping in 1895. The production from each of these great properties has ranged from 100,000 to 500,000 tons a year. At the time of visit these two properties were the only ones shipping ore. In 1913, the Juragua mines produced 369,213 long tons, the Daiquiri mines 591,322 long tons, and the El Cuero mines 130,183 long tons, a total of 1,090,718 tons.

The total production of the Juragua mines is 6,679,617 tons and that of the Daiquiri mines 6,744,004 tons.²

Situation of Daiquiri Mines

The Daiquiri mines are best reached by steamer from Santiago, the trip affording a beautiful view of the shelving coral deposits of the shore line, the intervening low lands, and the imposing front of the range continuous from Sierra Boniato to far east of Daiquiri. At the open roadstead there are a wharf and large ore bins. A railroad extends from the landing place up to the mines, a distance of 4 miles along the Magdalena River, a small water course here flowing through a moderately hilly country. A short distance below the mines the road leaves the river and follows Daiquiri Creek up to the town of the same name, where, at an elevation of 300 ft., the offices of the company are located. The

² *Mineral Resources of the United States*, 1913, vol. i, p. 317 (1914).

town lies in a basin-like valley, hills rising 500 to 800 ft. above it. Back of these, steep slopes lead up to the main range.

Interest naturally centers in Lola Hill, which attains a height of 600 ft. above the offices and is located on the ridge separating Magdalena River from Daiquiri Creek, for the principal mines are situated on the top and sides of this eminence. Smaller deposits lie east of the offices and extend irregularly toward Berraco, but little ore is shipped from these places and little definite geological evidence is obtained from them compared to that afforded by the magnificent exposures on Lola Hill.

Geology of Daiquiri

Granite.—A granitic rock with white rounded outcrops forms the most prominent part of the hills at the wharf and extends along the railroad for about 1 or 2 miles inland. The seaward side of the granite hills, which may be 150 ft. high, is covered by recent coral rock in three heavy benches. The granite is somewhat porphyritic and contains abundant phenocrysts of albite, while those of orthoclase, green hornblende, and quartz are less plentiful. The coarse groundmass is allotriomorphic and contains feldspar, quartz, biotite, and small cubes of magnetite. Epidote, chlorite, sericite, and kaolin are secondary products.

Diorite.—The contact between granite and diorite was not carefully studied, but it is probable that the two rocks are not far apart in time of origin.

The diorite continues up to the mines and is the predominant country rock of the district. It is usually light green in color when fresh, weathers in irregular, jagged, whitish outcrops, is traversed by irregular joint systems, and contains in places veinlets and masses of epidote, especially near the iron deposit. Darker varieties often form ill-defined masses in the normal rock, which is also often cut by white aplite dikes.

A typical diorite from the east side of the orebody, in the second open cut of the San Antonio mine, is of medium grain and distinctly porphyritic, this being especially noticeable in thin section. Under the microscope the most numerous phenocrysts prove to be of plagioclase and are rarely more than 1 or 2 mm. in length. Many of them are rounded or broken and poorly twinned. The few exact measurements which could be made indicate an oligoclase-andesine. The refraction, accurately determined, is from 1.535 to 1.545, which corresponds well to this mineral.

There are also one or two anhedral, but not corroded, phenocrysts of quartz, as well as several imperfect prisms and granular aggregates of colorless augite.

Sheaves of secondary green hornblende are abundant and needles of

the same penetrate the feldspar, all of this hornblende being probably derived from augite. Small grains of black ilmenite are fairly common, but most of them have been converted to titanite. In another rock of the kind, this process has been carried so far that no black iron ore remains.

The groundmass is microgranular, very interlocking and poikilitic or micro-porphyrific in places. It consists of grains of unstriated feldspar with index of refraction smaller than quartz, intergrown as indicated with quartz.

The secondary constituents comprise, besides hornblende, some epidote and a little sericite, the latter contained in the feldspar; also a little chlorite and calcite.

A partial analysis of this rock, made by C. S. Venable, of the Massachusetts Institute of Technology, yielded 64.07 per cent. SiO_2 , 5.62 per cent. CaO , 5.6 per cent. Na_2O , and 0.7 per cent. K_2O . The rock is thus more correctly defined as a normal quartz-augite-diorite-porphyry. The percentages of CaO and Na_2O correspond well with what might be expected and the small amount of K_2O is noteworthy, showing the rock to contain very little orthoclase. The percentage of black iron ore present is no more than would be expected in a rock of this kind.

Aplite.—White aplite dikes of irregular outline are found in places in the diorite. One of these occurs on the ridge between Magdalena River and Daiquiri Creek, $\frac{1}{4}$ mile north of Lime Point. It is a dirty-white medium-grained rock speckled with little grains of hornblende. The essential minerals are oligoclase, quartz, and orthoclase with a little hornblende, apatite, and titanite. The rock contains numerous feldspar phenocrysts, usually with excellent crystal outlines, and in places the texture of the coarse groundmass may be described as allotriomorphic. Micropegmatitic intergrowths of quartz and orthoclase are, however, very abundant.

Crystalline Limestone.—Besides the dikes mentioned, and the deposits of iron ore, the diorite includes small masses of limestone. The best locality is Lime Point, described below, but limestone is also found near the Barcelona mine on the west side of the Magdalena River and 2 miles from Lola Hill. Here, however, the field relations cannot be made out with certainty. Another small limestone mass is reported from the steep slope of the main range north of Daiquiri, and several others are found in close connection with ore in the Juragua mines.

Lime Point is situated 1,600 ft. southwest of the mine offices on the summit of the sharp ridge between Magdalena River and Daiquiri Creek. Here is a small mass of limestone almost 100 ft. in diameter and wholly inclosed in diorite, though the contacts are not well exposed. The stratification, trending north-south, is excellently shown, the beds being 1 to 2 ft. thick, with vertical dip. The rock is greenish, mottled and

crystalline, with medium grain. It is irregularly replaced by yellow garnet, a brownish green rhombic pyroxene, and some iron ore. In fact, some ore was taken from this place, but being of low grade was not shipped. A polished section showed that this iron ore consisted, like that of Lola Hill, of magnetite partly oxidized to hematite. A tunnel driven 50 ft. below the summit on the Magdalena side failed to find any continuation in depth of this limestone mass.

The significance of this occurrence in relation to the origin of the iron ores will be referred to below. It is clearly a partly contact-metamorphosed inclusion of a sedimentary series older than the intrusive diorite.

The Ore Deposit of Lola Hill

General Occurrence.—The great ore deposit of Lola Hill, from which 6,000,000 tons of high-grade iron ore have been mined, and in which much still remains, forms, broadly speaking, three lenses, partly connecting, aligned north-south, and aggregating 2,500 ft. in length. The dip is very steep to the east; the width varies up to 120 ft. Cutting squarely across the summit of Lola Hill, 600 ft. above the offices in the valley, it has been mined chiefly from a series of open cuts beginning 200 ft. above the base. Seen from a distance the hill presents a striking appearance, with its numerous benches with railroad tracks and steam shovels, and long lines of horizontal dumps where was deposited the barren rock which necessarily had to be removed from the walls of the deposits. From south to north the lenses are known as the San Antonio, nearest to the offices, the Lola, and the Magdalena. Three inclines lead down to the railroad from the open cuts and the San Antonio tunnel opens the lowest parts of the deposits from a few feet above the railroad level. This tunnel is still in magnetite ore, but the width is here only about 30 ft.

The Magdalena lens is developed by an open cut and by a tunnel approximately 200 ft. higher than the main railroad track at the offices.

Ore and Gangue.—Most of the ore consists of a dense or fine-grained mixture of magnetite and hematite, though in the lowest tunnels of the San Antonio and the Magdalena mines only magnetite occurs and this is rather coarser grained. The gangue consists of quartz, which is far more conspicuous in thin section than in the specimen. Garnet and a little amphibole, epidote, calcite, and pyrite are present in varying amounts. The ore in the Magdalena tunnel, for instance, is admixed in places with so much pyrite as to render it valueless for shipping. Copper stains are frequently seen and some parcels of oxidized copper ore have been shipped to the Cobre mine. Fresh chalcopyrite is rarely observed.

Recent analyses of iron ores from Daiquiri run as follows, in percentages:

Average of 134,000 Tons of Ore from the Lola Mine

Fe.....	59.9
P.....	0.019
SiO ₂	9.5
S.....	0.33

Average of 153,000 Tons of Ore from the Magdalena Mine

Fe.....	59.2
P.....	0.021
SiO ₂	11.3
S.....	0.89

Average of 42,000 Tons of Ore from the Barcelona Mine

Fe.....	58.7
P.....	0.03
SiO ₂	11.1
S.....	0.4

Earlier analyses quoted by Spencer³ show that the ore then mined averaged from 62 to 65 per cent. of iron and contained somewhat less silica. Analyses of average samples given by him run as follows:

Analyses of Average Samples of Iron Ores from the Lola Mine

	1896	1897
Fe.....	63.050	63.100
Mn.....	0.062	0.097
Cu.....	0.016	0.056
Ti.....	0.007	
S.....	0.048	0.072
P.....	0.025	0.029
Al ₂ O ₃	0.821	0.712
CaO.....	0.890	1.060
MgO.....	0.259	0.381
SiO ₂	7.585	7.225

Geological Field Relations of the Ore.—The ore, then, forms in general a tabular body in diorite, 2,500 ft. long, up to 100 or 120 ft. wide and at least 600 ft. deep. Actually it is divided in three parts by the intrusive rock and, moreover, as noted already by A. C. Spencer,⁴ it is cut in many places by igneous dikes. Two of these were observed in the open cuts of the San Antonio mine. Both were decomposed, but one of them which was 6 ft. wide proved to be a granite porphyry having phenocrysts of albite-oligoclase and quartz in a microgranular groundmass with feldspar and quartz sometimes in micropegmatitic intergrowths. The rock is evidently allied to the aplites and may be considered as later than the normal diorite.

³ *Report on a Geological Reconnaissance of Cuba*, p. 77 (Washington, 1901).

⁴ *Idem*, Fig. 12.

The exposures along the successive benches for a vertical distance of 400 ft. are excellent and show no direct evidence of surface oxidation in the way of limonite or kaolin. Each face is from 50 to 200 ft. below the original surface; the summit of the hill has been removed by the open-cut work.

In the main, the contacts between ore and the surrounding diorite are definite and the black orebodies stand out sharply against the whitish or greenish igneous rock. In detail, however, there is evidence of contact zones and intimate intermingling.

The iron ore itself is not so uniform as it appears at first glance. Generally fine grained, steel gray and giving a red streak, there are places where the admixture with quartz and pyrite becomes prominent; the quartz in irregular masses and veinlets, the pyrite in streaks of small crystals. Calcite is fairly abundant in places, but generally fills small irregular cavities or forms later crusts. More prominent than these minerals is garnet, which occurs abundantly in the ore, sometimes intimately mixed, but more often as heavy bodies or flat masses several feet in thickness. It is yellowish brown and massive, though showing in places small crystals in drusy cavities. A qualitative analysis showed it to be a normal andradite with little alumina.

The contact between ore and diorite can usually be located within 1 or 2 ft., but frequently dike-like masses of iron ore or of massive garnet 1 or 2 ft. wide project into the diorite for a distance of many feet, just as dikes of rock project into the ore.

The diorite adjoining the ore is rarely quite normal, but contains much epidote and is frequently darker and more fine grained than the typical rock, for a distance of 10 to 30 ft. from the contact. An occurrence of this kind from the lowest open cut on the San Antonio was designated in the field notes as "a heavy, fine-grained variety suggesting absorption of lime." Going west, the rock gradually becomes more normal, but the grain continues uneven and the rock contains much epidote. A specimen from this place about 10 ft. from the ore is further described below and designated as No. 7 Daiquiri.

In the deep tunnel, which extends from the railroad level into the San Antonio mine, the ore is only 30 ft. wide and consists of magnetite admixed with garnet, epidote, and vug fillings and coatings of calcite.

In the open cut of the Lola mine the exposures are similar to those of the San Antonio.

In the open cut of the Magdalena mine the ore is about 100 ft. wide and appears to be bounded by a fault on the east side. The ore is massive with but little admixture of gangue. The Magdalena mine is also opened by a tunnel about 200 ft. above the office, but on the slope toward the Magdalena River. Ore is here mined by underground methods. It is mostly coarse-grained magnetite oxidized to limonite

along fissures. No garnet was seen, but there is much pyrite in places and some of the ore is a dark-green mixture of amphibole, feldspar, quartz, and magnetite described below as No. 20 Daiquiri. A winze sunk 100 ft. below the Magdalena tunnel developed magnetite with 8 or 10 per cent. of pyrite.

Detailed Description of Ore and Gangue.—The "contact rock" (7 Daiquiri) previously mentioned was examined in some detail and proved a rather puzzling type. It is finely granular under the microscope and consists of 50 per cent. or more of amphibole and pyroxene, while the remaining half consists of a feldspathic mineral. It contains very few grains of iron ore, but a notable amount of small grains, crystals, and aggregates of titanite, which no doubt is derived from ilmenite.

There are a few equidimensional grains or rough prisms of colorless augite with normal extinction. Far more abundant is a pale-green uralitic hornblende in irregular grains or sheaves frayed at the edges, while needles of the same mineral are abundantly scattered through the feldspar, so that it would be impossible to separate the latter by heavy solutions. Apatite in small prisms is abundant. The feldspathic material consists of interlocking grains, which have low double refraction (about 0.004-5) and an index of refraction varying from 1.54 to 1.55, only one or two of these grains showing polysynthetic twinning with extinctions indicating andesine. The refraction points to oligoclase-andesine. The mineral is somewhat attacked by concentrated hydrochloric acid. A partial analysis was made of this rock by C. S. Venable and gave 51.67 per cent. SiO_2 , 11.54 per cent. CaO , 2.9 per cent. Na_2O , and 0.9 per cent. K_2O . This somewhat surprising result does not agree with the normal composition of an igneous rock composed of andesine, augite-hornblende, and suggests strongly the presence of a scapolite mineral. There is certainly feldspar in the rock, but the scarcity of twinning is remarkable and the refraction is somewhat too low for andesine. On the other hand, the section shows apparently only one prominent feldspathoid mineral; and the double refraction is decidedly too low for a calcium scapolite. The analysis confirms the suspicion entertained in the field that this dark "contact zone" has absorbed lime. Very likely the amphibole contains an exceptionally large percentage of calcium, or the feldspar is, in parts of the specimen, converted to a scapolitic mineral.

A specimen collected from the other, or eastern, contact shows a decided similarity to the one just described, although the contact rock on this side is not so well developed as on the other side. It is a greenish, fine-grained rock. In this section it shows an interlocking feldspar aggregate with some little quartz. Only one or two of the feldspar grains are twinned and give angles of extinction indicating andesine; the grains are in great part cloudy and filled with chlorite foils and horn-

blende needles. Fibrous hornblende is abundant in frayed prisms. There is no black iron ore, but some secondary titanite.

Much of the material mined is massive iron ore, and sections of this show little except a few included small grains of quartz, garnet, or pyrite. The most common gangue mineral is quartz, and this occurs intimately mixed with the iron ore. Analyses quoted on p. 2177 indicate an average of 10 or 11 per cent. of silica in the ore now being mined, and most of this silica is present as quartz. A section of typical quartzose ore is shown in Fig. 1. The specimen (18 Daiquiri) comes from the Magdalena open cut. The iron ore, which is magnetite with hematite rims,



FIG. 1.—PHOTOMICROGRAPH OF SILICEOUS ORE FROM MAGDALENA OPEN CUT. Black, Magnetite with some Hematite; gray, Quartz. Magnified 30 diameters.

forms irregular or rounded masses often bordered by projecting crystal faces, as may be seen in the figure. The quartz, which is coarsely granular, fills the interstices and contains a few small prisms of apatite. In places the impression is that of magnetite shattered and filled with quartz veins. There is little doubt that the magnetite is the earlier mineral. The section contains a few cubes of pyrite, which is distinctly associated with the quartz, and is later than the magnetite.

Another type of quartz magnetite rock is shown in Fig. 2; the specimen (21 Daiquiri) was taken from the deep San Antonio tunnel. The magnetite here forms small crystals and striking dendritic groups, all

imbedded in granular quartz, which also contains a considerable number of minute prisms of apatite. From this section the distinct impression is gained that the magnetite crystallized in a yielding medium, which later consolidated to granular quartz.

The presence of andradite garnet has been mentioned, but this mineral is less commonly seen in the shipped ore, for it forms large "bars" and masses which are easily sorted out. The ore shipped now contains only about 1 per cent. CaO , but the body as a whole certainly contains 3 or 4 per cent. of this constituent, present as calcite and andradite. Sections of iron ore often show small inclosed grains of garnet

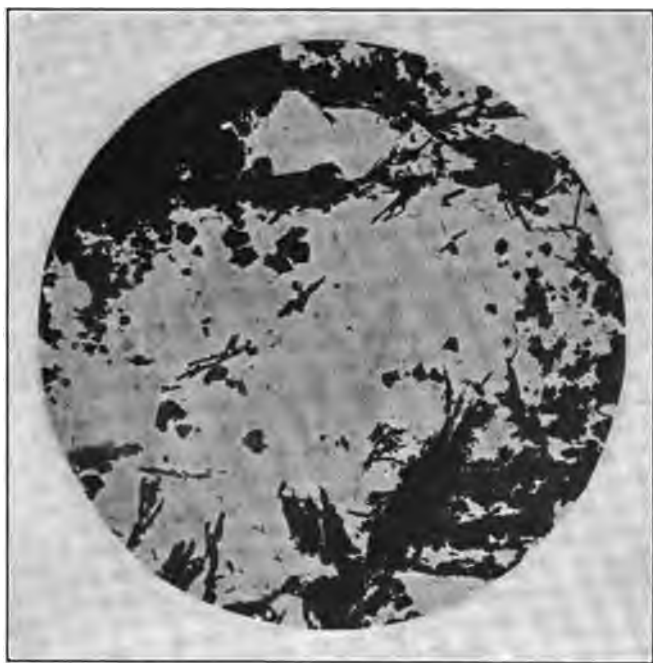


FIG. 2.—PHOTOMICROGRAPH OF SILICEOUS ORE FROM THE SAN ANTONIO TUNNEL.
Black, Magnetite; white, Quartz. Magnified 30 diameters.

and calcite, and both of these sometimes contain brownish green prisms of amphibole. The garnet rarely shows crystal outlines or the zonal birefringency so common in contact-metamorphic deposits. The calcite occurs either as grains in which, as stated, prisms of amphibole have developed, or as filling of small cavities, in which case it shows concentric deposition lines.

A type of lean ore which is rare at Daiquiri, but not uncommon at Juragua, is shown in Fig. 3. The specimen (24 Daiquiri) was taken from the Magdalena tunnel at the north end of the ore zone and occurs in heavy magnetite. The rock is coarse grained and dark green, and

under the microscope proves to be a granular mixture of monoclinic pyroxene, magnetite, quartz, and feldspar. The magnetite forms small crystals and rectangular bars, which by their arrangement indicate a coarse dendritic texture. The magnetite seems to be the earliest mineral. There is much granular or roughly prismatic augite, also some quartz, and a few grains of andesine feldspar, one of which is shown in the figure. While it is possible that this is an igneous rock, its interpretation as a replacement product of limestone would seem more probable.

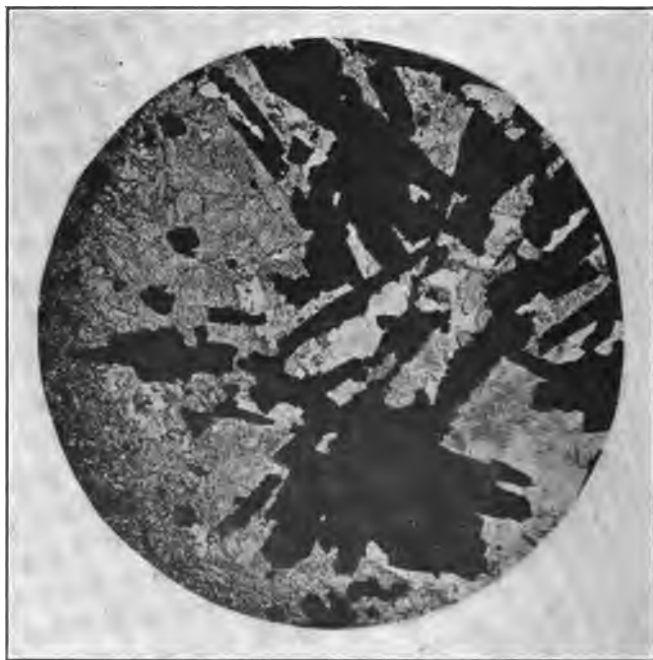


FIG. 3.—PHOTOMICROGRAPH OF LEAN ORE FROM MAGDALENA TUNNEL. Black, Magnetite; gray with granular surface, Augite; light gray, Quartz, with one grain of Andesine in center. Magnified 30 diameters.

Metallographic Examination.—Polished faces of the ore are difficult to prepare by reason of pitting and failure to take perfect polish. The sections studied all showed magnetite and hematite and the same relationship between the two minerals. Etching with hydrochloric acid accentuates the difference between magnetite and hematite a little better, both being slightly attacked, the latter more than the former. The magnetite, which predominates in most cases, has a dark-gray color with a tinge of red. The etching often brings out a system of lines, or two systems crossing at, say, 60° . These lines are caused by the octahedral parting in magnetite. The hematite is brighter and light gray in color and the polished faces are full of small pits.

The hematite occurs as a border of varying width around the magnetite and the contact between the two is usually wavy and irregular. Tongues of hematite, sometimes highly irregular in outline, enter along all possible openings, such as cracks and contacts between grains. The hematite has also a strong tendency to penetrate the magnetite along its parting planes, and in several places the parting is clearly brought out by the bright thin lines of hematite outlining a series of rhombs. In other sections the magnetite grains are penetrated by an exceedingly delicate, irregular network of hematite visible only with the highest magnification.

The pits in the hematite are probably caused by the local development of a softer or earthy facies of the mineral.

All gradations are present in the ores. Some are mostly magnetite, with very little hematite covering the grains as a thin film or penetrating them as a delicate filigree work. Others consist of about equal amounts of both minerals, and finally some appear as scaly hematite which are not magnetic and contain only a few residual grains of magnetite.

Very little limonite is present and, when found, bears no relation to the amount of hematite. In fact, the hematite appears to become hydrated only with great difficulty and most of the limonite seen, which came from the Magdalena tunnel, is a direct product of oxidation of magnetite.

The metallographic microscope shows that the pyrite is distinctly later than the magnetite; it usually forms cubes from 1 to 8 mm. in diameter. In many sections, which show mostly hematite, pyrite crystals remain fresh without any oxidation.

Genesis of the Deposits

From the above it is clear that the primary iron oxide of the deposits at Daiquiri is a magnetite, which subsequently has been altered more or less completely to a hematite. The discussion of genesis may therefore be divided into two parts: The first relating to the origin of the magnetite; the second to its subsequent hematitization.

The mineral association of the magnetite, particularly the presence of much garnet, shows that it originated under high-temperature conditions.

On the other hand, the hematitization is probably a low-temperature process developing gradually under the influence of oxidizing atmospheric waters.

The principal previous contribution to the question of the origin of the Daiquiri ores comes from the pen of A. C. Spencer of the U. S. Geological Survey.⁵ He considers three views:

⁵ *Report on a Geological Reconnaissance of Cuba*, pp. 79 to 82 (Washington, 1901).

1. The orebodies are an igneous product of processes of differentiation in a molten diorite magma.

2. The orebodies are a product of contact metamorphism exerted by the diorite on included masses of limestone.

3. The orebodies form a part of an older limestone-schist series, which was disrupted by the intrusion of the diorite. In this case the orebodies were formed by metamorphic processes in these sedimentary rocks long before the intrusion of the diorite.

Spencer believes the first and second theories are not very plausible, but is inclined to accept the third. He says:⁴

"It is necessary to suppose that the orebodies of the metamorphic series were originally widely distributed and of immense size. . . . A mass of many million tons weight floated upward by the buoyant effect of molten rock in motion from the interior towards the surface of the earth, is the only conception which adequately accounts for the mode of occurrence of the orebodies of the Magdalena and Lola mines at Daiquiri; while, though less strikingly shown, at Firmeza (Juragua) it is likely that the masses of schist, marble and ore have been likewise actually suspended in the molten lava."

Regarding first the theory of origin by differentiation in the diorite magma, we fully agree with Spencer that there are weighty objections against it. The association of ore with andradite, the lack of transitions between ore and diorite, the very small quantity of phosphorus present, and, last but not least, the definite character of the orebodies of *inclusions*—all these considerations militate strongly against this theory.

The third theory of Spencer presupposes a series of limestone and hornblende and epidote schists, which contained immense masses of magnetite, probably formed by regional metamorphic processes, and of which the present masses are but small remnants. The assumption of such a series of metamorphic rocks does not seem fully proved. At Daiquiri there is bedded limestone, but no schist whatever, and from Spencer's description of Juragua it does not clearly appear that the "schist" at that place really constitutes a conformable series with the limestone. The formation of such tremendous masses of magnetite, as must be assumed to have existed, by regional metamorphic processes is wholly improbable; we believe no such deposits are known. If they were formed by igneous metamorphism there seems to be no evidence of the old pre-dioritic intrusive which caused the orebodies to form. For these reasons it seems necessary to reject the theory of an older iron-bearing series of sediments.

There remains the second theory, accounting for the large masses of magnetite included in the diorite by contact-metamorphic action of the magma on an older limestone, by which the latter has become almost entirely replaced by magnetite derived from magmatic emana-

⁴ *Idem*, p. 82.

tions rich in iron. While it is freely admitted that the genesis of the Daiquiri deposits is difficult to explain, it will be shown that the view outlined in the previous sentence has much in its favor.

In the first place, the relations at Lime Point prove that a bedded limestone formation existed here before the intrusion of the diorite and that the fragment preserved at this place has been partly metamorphosed by the diorite, with the development of garnet, pyroxene, and magnetite, the latter in not inconsiderable quantities. The composition of the main orebody is that of a contact-metamorphic deposit with much andradite, garnet, quartz, and magnetite and some residuary calcite. It has been shown that grains of residuary calcite inclosed in magnetite contain prisms of amphibole, indicating exposure of these grains to metamorphosing influences. The only puzzling question is, why larger masses of calcite have not escaped the metamorphism, for in most deposits of this kind the change to ore is rarely complete. It may also seem strange that a smaller mass like that of Lime Point should have been partly metamorphosed, while the much larger mass of Lola Hill suffered almost complete replacement, but the capricious character of such metamorphism is well known.

The chemical composition of the ore is exactly that which would be expected in a contact-metamorphic deposit. The percentage of phosphorus is very low, while there is much pyrite in places and some chalcopyrite. The little phosphorus present is contained in the apatite prisms, which are inclosed in the quartz, and this apatite is believed to indicate that the quartz is not an infiltration which took place at low temperature, but that it was of magmatic origin and deposited from very hot solutions. Still another argument in favor of a genesis by replacement of limestone is furnished by the abnormal percentage of lime in the dioritic contact rock adjoining the deposit in many places. This percentage is much too high for a normal quartz diorite, or in fact for any igneous rock of the general type indicated by the thin sections, and points without doubt to a migration of lime outward into the igneous rock.

The enormous amount of calcium and carbon dioxide which must have been carried away and the correspondingly large additions of iron oxide, silica, and sulphur almost strain the imagination, it must be confessed, but students of contact metamorphism are well acquainted with similar and well-authenticated examples elsewhere. That there must have been a most energetic interchange of constituents between the limestone and the igneous rock is indicated by phenomena such as dike-like masses of andradite and magnetite along the contacts of the orebody. The dikes of igneous rock intruded into the iron ore appear to be somewhat different from the diorite and are either granite porphyry or aplite; that is, complementary dikes of a later generation.

The violent interchange of constituents along the contacts suggests that the diorite was fluid when the metamorphism took place.

Regarding the oxidation of the magnetite to hematite there is but little to add. Hematite appears to be more abundant close to the surface than in depth, hence an increased probability that descending surface waters effected this work. The change from magnetite to hematite involves but a slight addition of oxygen. It is certain that the amount of this gas was not large enough to oxidize appreciably the pyrite at a distance of about 100 ft. below the surface. The occurrence of fresh pyrite in almost completely hematitized magnetite is an interesting feature that deserves further investigation.

In explaining this mode of oxidation, the rapid erosion of this region should be borne in mind. At the surface, magnetite probably tends to become converted to limonite. It is likely that the conversion to hematite takes place at greater depth, below the water level, where the amount of oxygen available for the process is much smaller. According to this view the hematitization of this deposit took place at a depth of several hundred feet below the original surface, and as the conversion of hematite to limonite is exceedingly slow, the hematite acted as a protecting cover for the magnetite when erosion had lowered the surface and had exposed the hematitized portion of the deposit to the action of surface waters richer in oxygen.

The Ores of the Juragua Mines

General Features.—Unfortunately, time did not permit a visit to the Juragua mines, situated about 8 miles west of Daiquiri and connected by a railroad with the shipping point in Santiago Bay. From the available descriptions by Kimball and Spencer the deposits are on the whole very similar to those at Daiquiri, and mainly form irregular masses inclosed in an igneous intrusive rock. Spencer mentions a schistose rock closely associated with the iron ore at Firmeza mine and also describes crystalline limestone in close contact with the ore. The magnetite, according to a figure in his paper, apparently cuts across the limestone and both are intersected by narrow porphyry dikes. The present manager of the Juragua mines, De B. Whitaker, stated, in a personal interview, that limestone is found in at least two of the Juragua mines and the garnet also occurs in the ore. The deeper levels are said to yield chiefly magnetite.

A series of specimens was collected from the large stock pile at the shore of Santiago Bay and was examined with some care. No garnet or limestone was found in the stock pile. The ore is in large part the same magnetite, partly converted into hematite, which characterizes the Daiquiri deposits, but there was also more or less granular magnetite closely associated with dark-green fine-grained igneous rocks. Many

specimens consisted of a mixture of magnetite and actinolite or of magnetite and pyroxene. Epidote is abundant, mostly as veinlets distinctly later than the ore; intersecting the ore are also many veinlets of quartz and small dikelets of a quartzose granular rock.

The Juragua mines, which have been worked since 1884, have yielded about 6,000,000 tons of iron ore.

Detailed Descriptions.—The igneous rock mentioned above is fine grained and dark green. The thin section shows it to consist mainly of a basic feldspar, mostly labradorite in larger and smaller prisms and shredded prisms and needles of green hornblende. The rock is probably

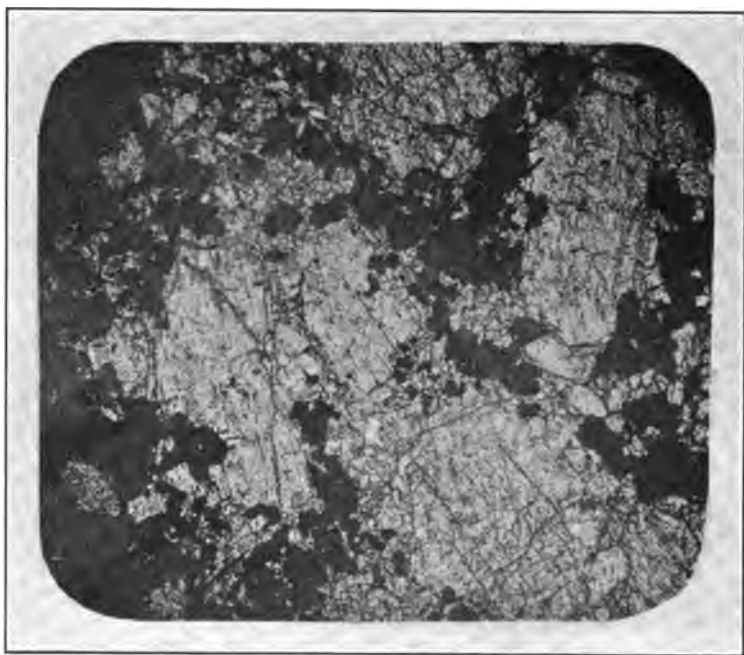


FIG. 4.—PHOTOMICROGRAPH OF LEAN ORE, JURAGUA MINES.

Black, Magnetite; gray with granular surface, Augite. Magnified 70 diameters.

an altered diabase porphyry. Accessories are magnetite in small grains, apatite, and quartz. Calcite and epidote form the secondary products.

Other dark-green rocks differ radically from the specimen just described and recall certain products of contact metamorphism. The principal constituent in these is augite in grains (Fig. 4), often small and rounded or irregular, more rarely, roughly prismatic. The other constituent is magnetite in irregular grains, apparently earlier in origin than the augite, which penetrates it in veinlets. In one section a single grain of a basic feldspar, probably bytownite, was found, while in another section feldspar was a little more abundant and none of the lamellæ showed

extinction of more than 21° , which would indicate andesine-labradorite. Epidote and quartz are secondary and only sparingly present.

Other specimens of dark-green lean ore contain much actinolite in bands or bladed aggregates, and more or less quartz in granular masses. Calcite is frequently present in small grains and veinlets or more rarely in larger cleavable masses about $\frac{1}{2}$ in. in diameter. Epidote is fairly abundant in larger masses, grains, and veinlets.

A number of ore specimens were cut by sharply defined veinlets of



FIG. 5.—PHOTOGRAPH OF BANDED MAGNETITE ORE, JURAGUA MINES. Black, Magnetite; light gray, Actinolite with some Quartz. Natural size. Specimen polished on two sides.

quartz with more or less perfect comb structure and frequently a central band of epidote or calcite; in one case the quartz veinlet contained well-formed crystals of magnetite and pyrite. Pyrite forms small granular streaks in some iron ore and occasionally a little chalcopryite with malachite is encountered.

One specimen of iron ore contained several dikelets apparently of igneous origin and from 2 to 10 mm. wide. They consist of a granular mass of pink feldspar, quartz, and scattered grains of magnetite.

A peculiar banded ore may deserve special mention. Its appearance

is shown in Fig. 5, which is a photograph in natural size of a specimen polished on two sides. The dark bands are magnetite, the lighter bands mainly actinolite, the two separated by curving but sharp boundaries. The needles of actinolite are mostly perpendicular to the banding. In some of the wider bands the actinolite is partly converted into asbestos. A thin section shows that much quartz in fair-sized, interlocking grains is associated with the actinolite and that some quartz is also present in the magnetite bands. The magnetite forms small irregular grains and aggregates; in places, however, rectangular outlines are seen, suggesting crystal form. The texture of this specimen strongly suggests diffusion banding, such as might form in a material, a limestone, perhaps, freely penetrated by hot iron-bearing solution. Somewhat similar textures have, we believe, been described from the limestone and "skarn" ores of central Sweden. Some of the iron ore from Juragua is distinctly reddish and scaly like normal hematite or seamed by red hematite. Most of it is, however, dark gray and fine grained and shows a red streak when scratched. The granular magnetite is less common.

Under the microscope quartz is seen to be a common constituent. Some of the iron ore forms rounded grains which are intimately intergrown with the interlocking quartz. Much of it, however, is in the form of elongated rectangles and needles crossing each other and the quartz grains in all directions or gathered into radiating clusters or sheaves.

The metallographic microscope shows that the Juragua ores, like those of Daiquiri, are intricate mixtures of primary magnetite with secondary hematite.

A specimen before polishing was a fine, black, granular mass, with a few bright surfaces of micaceous hematite, and streaks and irregular patches of granular pyrite, some of which showed striated crystal faces. The black, granular material is magnetic, although not as strongly so as pure magnetite, and gives a red streak. The study of the polished face shows that the greater part of the ore is granular magnetite. Hematite occurs as a rim around the magnetite grains, and penetrates them in an exceedingly intricate manner. Sometimes, where no visible opening exists, it penetrates as vein-like threads forming a beautiful, lace-like pattern.

Probable Genesis.—The iron ores of Juragua are then partly hematitized magnetites with gangue of quartz, augite, actinolite, and epidote. Igneous rocks of the diabase porphyry type occur in the district. The relationship of the magnetite to the augite and actinolite strongly suggests a contact-metamorphic origin of the ore and this conclusion is strengthened by the presence of chalcopyrite in the ore.

Literature of the Daiquiri Mines

- JAMES P. KIMBALL: The Iron Ore Range of the Santiago District of Cuba, *Trans.*, xiii, pp. 613 to 634 (1884-85).
- : Geological Relations and Genesis of the Specular Iron Ores of Santiago de Cuba, *American Journal of Science*, Series 3, vol. xxviii, pp. 416 to 429 (1884).
- F. F. CHISHOLM: Iron-ore Beds in the Province of Santiago, Cuba, *Proceedings of the Colorado Scientific Society*, vol. iii, pp. 259 to 263 (1888-90).
- C. WILLARD HAYES, T. WAYLAND VAUGHAN, and ARTHUR C. SPENCER: *Report on a Geological Reconnaissance of Cuba*, made under the direction of General Leonard Wood, Military Governor, pp. 69 to 83 (Washington, 1901).
- ARTHUR C. SPENCER: The Iron Ores of Santiago, Cuba, *Engineering and Mining Journal*, vol. lxxii, pp. 633 to 634 (Nov. 16, 1901).
- : The Mayari and Daiquiri Iron Ore Mines. Articles published in *The Iron Age*, vol. lxxx, pp. 421 to 426 (Aug. 15, 1907) and vol. lxxxi, pp. 1149 to 1157 (April 9, 1908).

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INDUSTRIAL SECTION

Weiss Transit and Level

The illustrations shown are the transits and levels manufactured by the Weiss Instrument Co. of Denver, Colo., and they state that city and mining engineers are securing satisfactory results.

It is claimed that their patented prismatic telescope gives increased illumination with an astronomical eye-piece. The prisms are set in a milled seat for this purpose, and cemented in so they can never come loose or out of adjustment. All adjustment for collimation is done with the cross-hairs. This telescope is 8 in. long for magnifying power of 20 diameters against 12 in. for regular old style. The field is $1^{\circ} 45'$ with a clear aperture of $1\frac{1}{8}$ in. The standards are $4\frac{3}{4}$ in. high from plate to center of telescope axis. Weight 10 lb.

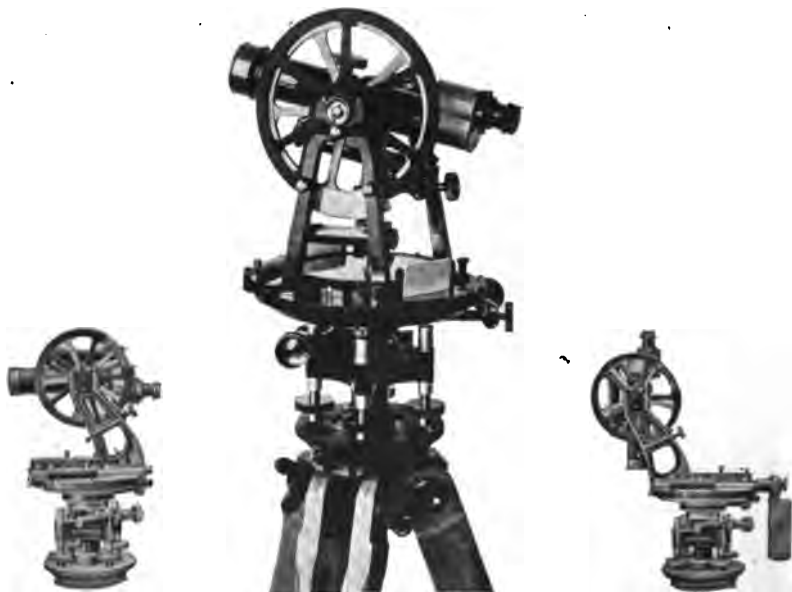


FIG. 2.

FIG. 1.

FIG. 3.

The Light Mountain Transit of the same type, Fig. 1, has a telescope $6\frac{1}{2}$ in. long, magnifying power 17 diameters; field $2^{\circ} 15'$; clear aperture $1\frac{1}{16}$ in.; weight 9 lb.

The Mine Transit made by this company is interesting because it may be sighted directly down a shaft. The principal advantage claimed for this type is the arrangement by means of which vertical sights can be had without an additional telescope. As illustrated in Fig. 2 the telescope is in the position of a regular transit, and used exactly the same.

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It is our intention to make the advertising section of the *Bulletin* as comprehensible as possible, so that members can turn to it for complete information as to where to buy the best mining and metallurgical supplies at the lowest prices. No firms or supplies of dubious reputation are advertised in the *Bulletin*. If members, in buying any material, will refer to the advertisement of that material in the *Bulletin*, or if they do not find it advertised in the *Bulletin*, will, when ordering, ask the manufacturers why the goods were not advertised in the Institute publication, it will be of material benefit to all of us, because the chief reason why advertisers are unable to trace direct results is because purchasers do not mention the *Bulletin* when writing. Finally, if members, when in need of anything whatsoever, will write to the Institute headquarters, an effort will be made to send them a complete list of manufacturers able to supply the desired materials of good quality. The furnishing of these lists will be a benefit to our Advertising Department, as well as to the members. We need scarcely call attention to the advantages obtained by the members from the money which advertising brings in, because all this money is returned directly to the members in one form or another.

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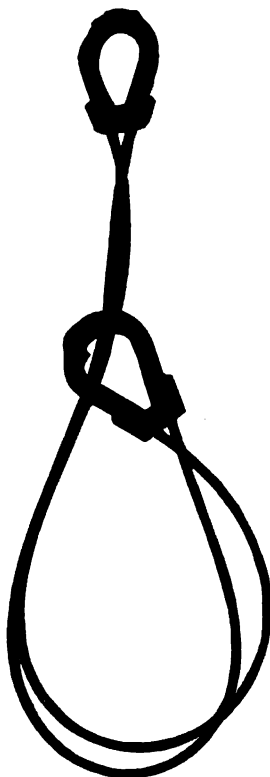
Zinc Ores, Carbonates, Sulphides and Mixed Ores, Copper Ores, Copper Matte, Copper Bullion, Lead Bullion, Lead Ores, Antimony Ores, Iron and Manganese Ores, Copper, Spelter, Antimony, Antimonial Lead, Sulphate of Copper, Arsenic, Zinc Dust.

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Fig. 3, the standards and telescope are moved by means of a fast screw along precisely fitted key ways, by means of which collimation is retained the entire distance. In using it as illustrated in Fig. 3 there is a counterpoise which overcomes all overbalances of the telescope. It has edge graduations, vertical circle and verniers, horizontal circle reading to minutes, the regular head and shifting center. Phosphor bronze and bell metal are used throughout.

The Roebling Wire Rope Sling

The recent development of the Roebling Wire Rope Sling is said to be one of the greatest advances in the sling field. The handling of heavy machinery, machine parts, castings, structural parts, trucks, etc., has long been a problem when chain or manila slings only were obtainable.



ONE TYPE OF ROEBLING ANCHOR SLING

The uncertainty of chain or manila slings has resulted in many accidents—no one knowing the actual strength, owing to forgings in tension of chain, and the uncertainty of fiber structure of manila after a few months' hard service.



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TAPES



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Heretofore, the importance of the sling has been often overlooked, owing to the apparent simplicity of the equipment. Men in charge have not been inclined to give its design serious consideration.

The Roebling sling has no forgings in tension, thus being uniform in tensile strength. This sling is manufactured from the highest grade of steel—Roebling Blue Center—affording the highest factor of safety per size.

To-day, therefore, the doubt and uncertainty of the past in regard to tensile stress has been reduced to a minimum.

At present there are three types: The Anchor, Column, Bridle. As evidence that, size for size, the Roebling sling is of greater tensile strength than chain, besides having no forgings in tension, the Roebling Company gives the following interesting data on common chain and wire rope slings:

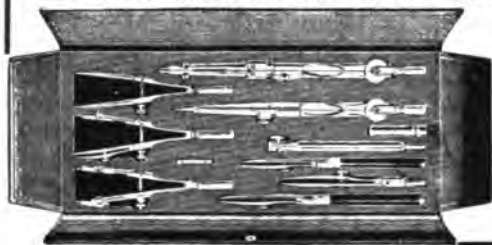
Working Load, Pounds	Size Slings			Breaking Strengths		Factor Safety		Comparison of Weights, 16 Ft. Slings			Diam. Chain Sling of Equivalent Strength to Anchor Sling	
	"2B," Inches	"3B," Inches	Diam. Anchor Sling, Inches	"3B" Chain Sling, Pounds	Anchor Sling, Pounds	"3B" Chain Sling	Anchor Sling	"2B" Chain, See note, Pounds	"3B" Chain, See note, Pounds	Anchor Sling, Pounds	Anchor Sling, Inches	"3B" Chain Sling, Inches
4,000	$\frac{5}{8}$	$\frac{5}{8}$	$\frac{1}{2}$	20,720	34,000	5.15	8.50	68	68	42	$\frac{1}{2}$	$1\frac{1}{4}$
6,000	$\frac{3}{4}$	$\frac{3}{4}$	$\frac{5}{8}$	30,240	54,000	5.04	9.00	95	95	60	$\frac{5}{8}$	1
8,000	$\frac{7}{8}$	$\frac{7}{8}$	$\frac{5}{8}$	40,880	54,000	5.11	6.75	135	135	60	$\frac{5}{8}$	1
10,000	1	1	$\frac{3}{4}$	53,760	76,000	5.37	7.60	180	180	72	$\frac{3}{4}$	$1\frac{1}{2}$
12,000	$1\frac{1}{8}$	$1\frac{1}{8}$	$\frac{3}{4}$	68,320	76,000	5.69	6.33	220	220	72	$\frac{3}{4}$	$1\frac{1}{2}$
16,000	$1\frac{1}{4}$	$1\frac{1}{4}$	$\frac{7}{8}$	84,000	100,000	5.25	6.25	280	280	95	$\frac{7}{8}$	$1\frac{1}{2}$
20,000	$1\frac{1}{2}$	$1\frac{3}{8}$	1	101,360	132,000	5.07	6.60	390	330	110	1	$1\frac{1}{2}$
25,000	$1\frac{5}{8}$	$1\frac{1}{2}$	$1\frac{1}{8}$	120,960	160,000	4.84	6.40	475	390	150	$1\frac{1}{8}$	$1\frac{3}{4}$
30,000	$1\frac{3}{4}$	$1\frac{5}{8}$	$1\frac{1}{4}$	140,000	174,000	4.67	5.80	540	475	165	$1\frac{1}{4}$	$1\frac{3}{4}$
35,000	2	$1\frac{7}{8}$	$1\frac{3}{8}$	175,000	206,000	5.00	5.90	725	660	272	$1\frac{3}{8}$	$2\frac{1}{4}$
40,000	$2\frac{1}{8}$	2	$1\frac{1}{2}$	198,000	234,000	4.95	5.85	825	725	282	$1\frac{1}{2}$	$2\frac{1}{4}$
45,000	$2\frac{1}{4}$	$2\frac{1}{8}$	$1\frac{3}{4}$	224,000	260,000	4.97	5.80	950	850	440	$1\frac{3}{4}$	$2\frac{3}{4}$
50,000	$2\frac{3}{8}$	$2\frac{1}{4}$	$1\frac{3}{4}$	250,000	300,000	5.00	6.00	1100	975	490	$1\frac{3}{4}$	$2\frac{3}{4}$

NOTE.—Chain slings with long link on one end and pear-shaped slip link on other end.

Trade Note

The Hauck Manufacturing Co., 142 Livingston Street, Brooklyn N. Y., announces that it has an interesting exhibit of portable and stationary oil-burning appliances and kerosene torches at the Panama-Pacific International Exposition in San Francisco. Its exhibit and demonstrating booth is located at Block No. 49, Machinery Hall. This should prove of interest to any of our members visiting the Fair.

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Electric Caps for Wet Work

With very few exceptions those who use explosives for blasting purposes have to contend with water in the bore holes to a greater or lesser extent. When blasting by electricity and there is very little water encountered, satisfactory results will be obtained by using the ordinary type of electric blasting caps. However, when a considerable quantity of water is found in the bore holes, it is more economical to use electric blasting caps that are made for this special condition. The reason for this is that the cotton insulation on the wires of an electric blasting cap only furnishes protection to the wires from abrasion and slight protection from water, the water easily penetrating the cotton insulation, causing short circuits and consequently misfires, which are not only expensive but also dangerous.

Because of the water conditions the Du Pont Powder Co. has placed on the market two kinds of electric blasting caps to meet the demands and also to furnish the consumer with an electric detonator which can be used under the most severe conditions. One kind is known as the Du Pont Waterproof Electric Blasting Cap and the other as the Du Pont Submarine Electric Blasting Cap.

The construction of these two kinds of electric blasting caps is exactly the same so far as concerns the detonating charges they contain, the wires, the cotton insulation on the wires and the filling material above the detonating charge. The difference will be found in the length of the copper shells, the insulation under the cotton insulation and the gutta percha covering the copper shells and extending about 2 in. up the wires on the Submarine type. The length of the copper shells of the No. 6 Waterproof and Submarine types is $2\frac{3}{4}$ in., of the No. 8 Waterproof and Submarine types is 3 in.

While the Waterproof and Submarine types cost more than the ordinary type, yet where water is encountered they are more economical and satisfactory to use, and users of blasting supplies should use them to obtain best results and eliminate the liability of misfires.

In quarrying, railroad blasting, tunnel blasting, shaft sinking and so on, where a depth of water not over 30 ft. is met with, the Waterproof type should be used. Where the water is over 30 ft. deep then the Submarine type should be used.

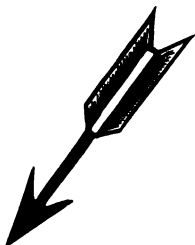
Waterproof and Submarine Electric Blasting Caps are made with copper wire only. The Waterproof can be secured with wire lengths as follows:

4	8	12	16	20	24	28 ft.
6	10	14	18	22	26	30 ft.

although longer length wires may be had if necessary.

The Submarine Electric Blasting Caps will not be carried in stock and orders will only be filled as received in lengths desired.—*Du Pont Magazine*.

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This booklet written by Mr. James Humes was made possible by the "distinctive features" of Traylor Automatic Crushing Rolls. It tells all about "The Fleeting Roll" feature which is only one of the many dependable points of superior roll construction.

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Asbestos

Fifty years ago there was practically no asbestos industry. Today the yearly output runs well up into the millions of dollars in America alone, while every civilized community in the world uses the mineral in various forms.

Although we have ample evidence that this material was known and used by the versatile Egyptians and their long-buried contemporaries, it was almost forgotten for many thousands of years, until in quite recent times the advent of high-pressure steam brought the need for a material which would resist higher temperatures than anything then in use.

Attention was therefore directed to the large deposits of asbestos known to exist in Canada, particularly in the eastern townships of Quebec Province. This region is today the principal source of supply for the world's consumption.

Asbestos possesses a faculty unique among minerals, that of being spun and woven into thread or cloth. It is also capable of being felted, but unlike all organic fibers, which are tubular in their formation and therefore easily twisted without breaking, asbestos is non-tubular, and for a long time this threatened to be a stumbling block in the way of its manufacture into textiles.

It is certain the ancients overcame this trouble by mixing a vegetable fiber with the asbestos, but for modern purposes this is impossible. The difficulty has now been successfully overcome by special machinery, so that asbestos is now spun into thread, twisted into cord of considerable strength, and woven into a variety of fabrics.

The list of possibilities of asbestos manufacture is far from being exhausted by textiles.

The engineer is indebted to it for packings for valves, pistons, and joints, non-conducting coverings for boilers, pipes, steam chests and all heated surfaces, brake blocks, stack and flue ducts and linings, and a variety of minor articles. Wherever heat shows a tendency to become unruly, asbestos can be depended upon to curb it.

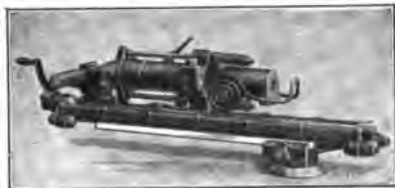
The electrician and electrical engineer owe a great deal to the introduction of asbestos because it is the base of many of the best non-conductors of electricity yet discovered.

More recently, under the name of "Transite" and "Transite Ebony," the H. W. Johns-Manville Co. have introduced two kinds of asbestos "wood" which is being largely used for switchboard purposes and a variety of other electrical uses, including station, street-car and railway paneling and flooring, telephone and motion-picture booths, etc., for which its ease of working, lightness, and fireproof qualities make it specially suited.

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The advance formerly obtained with two $3\frac{1}{4}$ " piston drills, using four men and working three eight hour shifts per 24 hour day, was 120 feet per month.

Reference: Draper Mine, Soulsbyville, Cal.

THE "JACKHAMER" IS DESCRIBED IN BULLETIN 4121

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with asbestos boards, shingles, and felts, while fireproof roofings have a basis of asbestos. Asbestos is used also as a siding for bungalows, garages, and other small buildings. Larger houses are plastered with asbestos stucco, roofed with asbestos shingles (which may also be used as siding), and may be painted with fireproof asbestos paint.

By combination with various asphalts and Portland or hydraulic cement, its field is greatly widened and a great number of additional products are added to the list.

The field for asbestos products is being constantly widened by modern industrial expansion. Its marvelous adaptability and its powers of resistance to heat, water, oil, and acids, combined with a capacity for almost indefinite wear, fully entitle it to claim the title of "the mineral of many marvels."

The H. W. Johns-Manville Co. was one of the earliest to recognize the immense future possibilities of asbestos, and from the first days of the industry this firm has been prominently identified with the growth and development of the modern demand for asbestos products. We are indebted to them for these brief outlines of the industry.

An Automatic Fire-Door Release

The Automatic Sprinkler Co. of America, 125 William Street, N. Y., calls attention to its device for automatically releasing fire doors. It is claimed that it is absolutely positive in its action, and operates on a new principle of fire protection. Normal changes of temperature have no effect on it. The manufacturer states that the fire door closes before a fire can get within 25 feet of the opening, and it can be arranged to sound an alarm if desired. The device seems to be simplicity itself, as all that is necessary for installing it is to fasten it to the wall, and extend and connect to it the cord of the door. The company will be glad to send full details to those interested.

Trade Catalogues

Recording Instruments.—The Bristol Company, Waterbury, Conn. Several bulletins and pamphlets on pressure and vacuum gauges, recording thermometers, electric pyrometers, time recorders and ammeters. Well written and profusely illustrated. Also booklet calling attention to their exhibit at the Panama-Pacific International Exposition, and the results of twenty-five years' experience in the manufacture of recording instruments.

Pulverized Fuel Combustion.—Aero Pulverizer Co., 63 Broadway, N. Y. Literature describes their method of pulverizing coal for boiler



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and metallurgical furnaces, and all furnaces requiring a temperature and quality of flame under prompt and easy control. Illustrated.

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Oxygen and Hydrogen Production.—International Oxygen Co., 117 Broadway, New York, 24 pp. Describes and illustrates the importance of pure oxygen for welding and cutting, and the saving realized by producing the gas on the user's own premises.

Temperature-Recording Instruments.—H. Weinlagen, 161 Leonard St., New York, 47 pp. A descriptive and illustrated catalogue on the line of Thermometers, Hydrometers, Pyrometers, Barometers and Recording Instruments manufactured, together with prices of the same. A useful catalogue for users of such instruments.

Hardness, Heat, etc., Measuring Instruments.—Shore Instrument Company, 557 W. 22d St., New York. Several descriptive and illustrated leaflets on their various apparatus for measuring materials for hardness, heat, elasticity, etc. Also a reprint from the *Iron Age* describing the Shore method of producing partial hardness of steel, with the idea of eliminating warping.

Rope.—The Waterbury Co., 88 South St., New York. Several well-illustrated and descriptive booklets and catalogues on the Waterbury Brand of wire, manila, armored, and fiber-clad rope, together with data and prices of interest to users of rope of all kinds.

The Venturi Meter.—The Builders Iron Foundry, Providence, R. I., 35 pp. A handsome and well-illustrated descriptive catalogue of their product for gravity mains, filtration plants, pump discharge lines, hydraulic plants, etc. Tables and other interesting data are included.

Diesel Engines.—The Busch-Sulzer Bros. Diesel Engine Co., St. Louis, Mo., 15 pp. Catalogue, handsomely illustrated, describes their type B engine, its efficiency and economy. Fuel control, life and manufacturing data are given.

Drill Sharpeners.—Sullivan Machinery Co., Chicago, Ill. Bulletin No. 72 just published, gives valuable information on the Sullivan Drill Sharpener for making drill bits and shanks by hammering. 23 pp., illustrated.

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ALPHABETICAL LIST OF ADVERTISERS

	Page
Albany Lubricating Co.	5
Allis-Chalmers Manufacturing Co.	16
Beer, Sondheimer & Co.	3
A. S. Cameron Steam Pump Works	15
Crocker-Wheeler Co.	16
Denver Fire Clay Co.	16
Denver Rock Drill Co.	16
Dorr Cyanide Machinery Co.	7
Du Pont Powder Co.	3
Edison Storage Battery Co.	Inside Back Cover
Ensign Bickford Co.	7
General Electric Co.	Outside Back Cover
Goodman Manufacturing Co.	17
B. F. Goodrich Co.	Inside Front Cover
W. & L. E. Gurley	1
Harbison-Walker Refractories Co.	18
Heller & Brightly	1
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Ingersoll-Rand Co.	11
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Lufkin Rule Co.	5
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Robins Conveying Belt Co.	19
John A. Roebling's Sons Co.	13
P. H. & F. M. Roots Co.	20
E. G. Ruehle & Co.	7
Chas. A. Schieren.	5
Sullivan Machinery Co.	11
Terry & Tench Co.	20
Traylor Engineering & Manufacturing Co.	9
Union Iron Works	1
L. Vogelstein & Co.	23
Vulcan Iron Works.	15
Westinghouse Electric & Mfg. Co.	21
Weston Electrical Instrument Co.	5



NOVEMBER, 1915



Bulletin of the American Institute of Mining Engineers

PUBLISHED MONTHLY

SECTION I

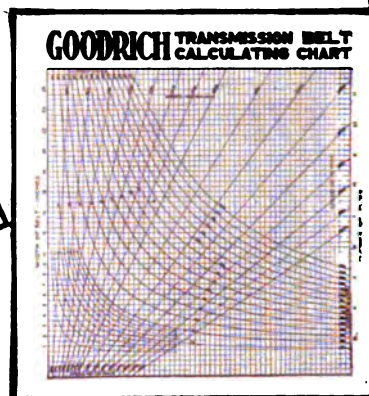
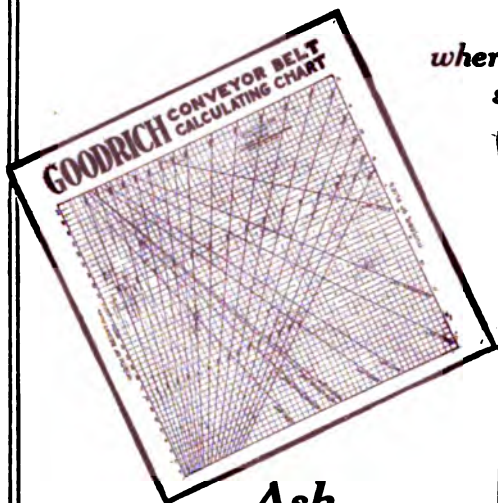
	PAGE		PAGE
Personal.....	iv	Library.....	vii
Nominations for Officers.....	v	Membership.....	x
Employment.....	v	Institute Committees.....	xxiii
Local Section News.....	vi	Officers and Directors.....	xxx

SECTION II.—PAPERS AND DISCUSSIONS

The Disseminated Copper Ores of Bingham Canyon, Utah. By J. J. Beeson	2191
Illumination of Mines. By Robert P. Burrows	2237
The Evolution of Drilling Rigs. By R. B. Woodworth	2247
Industrial Section	Interpolated in Advertising Section.

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[illegible]

*Present position*_____

Signature.

Dated _____ *191*

EXTRACTS FROM THE CONSTITUTION.

ARTICLE II.—MEMBERS.

SEC. 1. The membership of the Institute shall comprise four classes, namely: 1. Members; 2. Honorary Members; 3. Associates; 4. Junior Members. * * *

SEC. 2. The following classes of persons shall be eligible for membership in the Institute, namely : as Members, all professional mining engineers, geologists, metallurgists, or chemists, and all persons actively engaged in mining and metallurgical engineering, geology, or chemistry ; as Associates, all persons desirous of being connected with the Institute who in the opinion of the Board of Directors are suitable.

As Junior Members, all students in good standing in engineering schools who have not taken their degrees and who are nominated by at least two of their instructors. * * *

Every candidate for election as a Member, Associate, or Junior Member must be proposed for election by at least three Members or Associates, must be approved by the Committee on Membership, as prescribed in the By-Laws, and must be elected by the Board of Directors.

Harvard College Library

Aug. 14, 1916.

Bequest of

Erasmus Darwin Leavitt.

BULLETIN OF THE AMERICAN INSTITUTE OF MINING ENGINEERS

PUBLISHED MONTHLY

No. 107

NOVEMBER

1915

Published Monthly by the American Institute of Mining Engineers at 212-218 York St., York, Pa., H. A. WISOTZKEY, Publication Manager. Editorial Office, 29 West 39th St., New York, N. Y., BRADLEY STOUGHTON, Editor, F. O. PIERCE, Asst. Editor. Cable address, "Aime," Western Union Telegraph Code. Subscription (including postage), \$10 per annum; to members of the Institute, public libraries, educational institutions and technical societies, \$5 per annum. Single copies (including postage), \$1 each; to members of the Institute, public libraries, etc., 50 cents each.

Entered as Second Class matter January 28, 1914, at the Post Office at
York, Pennsylvania, under the Act of March 3, 1879.

NEW YORK MEETING, FEBRUARY, 1916

The 112th Meeting of the Institute, and the Annual Business Meeting, will be held at the Headquarters of the Institute in New York City, Feb. 15 to 17, 1916.

The Chairmen of several of the technical committees report that papers on subjects of timely interest in their respective branches have been promised. David H. Browne, who was appointed Chairman of the Committee on Arrangements, is organizing his committee for active work.

Members are urged to make their plans now to be present at this meeting to avail themselves of the pleasant opportunity for greeting old friends and making new ones among those engaged in the same profession.

All papers to be presented at this meeting of the Institute must be in the hands of the Secretary on or before Dec. 1, 1915.

PERSONAL

(Members are urged to send in for this column any notes of interest concerning themselves or their fellow-members.)

Members and guests who registered at Institute headquarters during the period Sept. 10 to Oct. 10, 1915:

George H. Blakemore, Sydney, N. S. W.
H. Haage, Sydney, N. S. W.
C. A. Filteau, New York, N. Y.
Myron R. Walker, Cerro de Pasco, Peru.
William R. Grunow, Morenci, Ariz.
E. DeGolyer, Norman, Okla.
William W. Elmer, Tuttletown, Cal.
Huntington Adams, Iquique, Chile.

W. B. A. Dingwall, San Antonio, Tex.
R. H. Jeffrey, London, England.
H. S. Emlaw, Spring Lake, Mich.
Y. T. Woo, Tientsin, China.
A. L. DeDeken, Louvain, Belgium.
K. P. Swensen, Tokyo, Japan.
Charles Enzian, Wilkes-Barre, Pa.
C. F. Schnepf, Cobalt, Ont., Canada.

W. Ray Cox, for several years Mineral Inspector of the General Land Office at Portland, Ore., has been appointed Professor of Mining and Metallurgy in the School of Mines, Oregon Agricultural College, Corvallis, Ore.

J. E. Johnson, Jr., has been elected Chairman of the New York Section of the Mining and Metallurgical Society of America; **A. H. Rogers** was re-elected Vice-Chairman.

Arthur W. Burgren has accepted a position with the Cinco Minas Co., at Magdalena, Jalisco, Mexico.

W. L. Honnold, who recently retired from the Chairmanship of the Brakpan Mines, the Springs mines, and the Transvaal Gold Trust, has taken up his residence at Hyde Park Hotel, Knightsbridge, London, England.

C. F. Kelley, Vice-President of the Anaconda Copper Mining Co., has transferred his residence from Butte to New York, but will continue to have direct connection with all operations of the company in Montana.

John Gross has opened an office as mining and metallurgical engineer at 523 McPhee Building, Denver, Colo.

H. P. Zeller, Assistant Superintendent of the coke plant of the Republic Iron & Steel Co., Youngstown, Ohio, has resigned to become Superintendent of the new byproduct plant of the Toledo Products Co.

Thomas A. Edison was elected Chairman; **William L. Saunders**, Second Vice-Chairman, and **Thomas Robins**, Secretary, of the Naval Consulting Board at a meeting held in Washington on Oct. 7, 1915.

L. W. Ledyard is Superintendent of the Teck-Hughes Gold Mines, Ltd., of Kirkland Lake, Ont., Canada.

Walter E. Gaby is Assistant Geologist on the staff of the Anaconda Copper Mining Co., Butte, Mont.

Corey C. Brayton has returned to San Francisco after a summer spent in making placer examinations in the Nome region, Alaska.

REPORT OF THE COMMITTEE OF NOMINATIONS

September 28, 1915.

The Nominating Committee of the American Institute of Mining Engineers has the honor to present the following nominations for officers of the Institute to be voted for at the 1915 Annual Meeting:

For President:	L. D. Ricketts.....	District 0
For Vice-Presidents:	Karl Eilers.....	District 0
	James MacNaughton.....	District 4
For Directors:	Thomas B. Stearns.....	District 7
	Charles F. Rand.....	District 0
	Charles W. Goodale.....	District 5
	Edwin Ludlow.....	District 2
	George D. Barron.....	District 0

F. W. BRADLEY, *Chairman.***ENGINEERS AVAILABLE**

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons introduced by members.)

Graduate mining engineer, with 10 years' experience, having an office in New York, would like to represent a mining company as selling and purchasing agent, and general supervision if desired. Machinery agency considered. No. 259.

Member, technical graduate, with several years' experience as engineer for a pyrites company, and as geologist and engineer for copper, nickel, and silver companies, desires a position as engineer or superintendent of mines. No. 260.

Member, technical graduate, aged 29, with experience as draftsman, surveyor, and superintendent of iron and gold mines, and as construction engineer, is open for a position as mine superintendent, or in mine examination or construction. No. 261.

Member, technical graduate, aged 32, single, with five years' experience as mining geologist, open for engagement as resident geologist or field engineer in mine examinations. No. 262.

Member, graduate of Columbia School of Mines, aged 45, with experience as mine surveyor, engineer, and superintendent, and as designing, manufacturing and constructing engineer; familiar with mining methods, with handling of men, with machinery, with designing, with erection and construction work. Available for long or short engagements in United States or Canada. Now in the West. No. 263.

Member, graduate engineer, aged 44, with over 22 years' experience in connection with the development of large coal and coke properties in different parts of the country, desires position as general manager or general superintendent of large bituminous coal producing, or will take an interest and the management of a smaller operation. No. 258.

SOUTHERN CALIFORNIA LOCAL SECTION*Executive Committee*SEELEY W. MUDD, *Chairman*C. COLCOCK JONES, *Vice-Chairman*FREDERICK J. H. MERRILL, *Secretary-Treasurer*, 631 Higgins Bldg.,
Los Angeles, Cal.

RALPH ARNOLD

A. B. CARPENTER

The Southern California Section held a dinner and meeting at the Sierra Madre Club, Los Angeles, on Friday evening, Sept. 28, 1915, at which 21 members and guests were present. The Vice-Chairman, C. Colcock Jones, presided, in the absence of the Chairman.

The amendment to the By-Laws providing for the increase of the Executive Committee from five to seven members, having been formally presented at the annual meeting of June 18, 1915, and duly announced for consideration at this meeting, was voted on and unanimously adopted.

Prof. Robert H. Richards of the Massachusetts Institute of Technology, made a short address on the subject of ore concentration.

The paper on the Geology and Ore Deposits of Mojave County, Arizona, prepared by F. C. Shrader, was presented to the meeting by the Secretary, and was discussed by Allen Burris and Dr. L. D. Godshall.

F. J. H. MERRILL, *Secretary*.

UTAH LOCAL SECTION*Executive Committee*C. W. WHITLEY, *Chairman*WALTER FITCH, *Vice-Chairman*ERNEST GAYFORD, *Secretary*, 159 Pierpont Ave., Salt Lake City

The annual meeting of the Utah Local Section of the Institute was held at Salt Lake City on Oct. 4, 1915. At this meeting papers were read by T. S. Carnahan and J. M. Callow.

Officers were elected for the ensuing year, as follows:

Chairman, C. W. Whitley.*Vice-Chairman*, Walter Fitch.*Secretary-Treasurer*, Ernest Gayford.

The membership of the Utah Local Section is now 108.

ERNEST GAYFORD, *Secretary*.

LIBRARY

AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS

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The Library of the above-named Societies is open from 9 A.M. to 10 P.M. on all week-days, except holidays, from September 1 to June 30, and from 9 A.M. to 6 P.M. during July and August. The Library contains about 55,000 volumes, including sets of technical periodicals and the publications of scientific and technical societies.

Members of the Institute, with few exceptions, are forced to spend a portion of their time in localities isolated from sources of information. To these the Library can render valuable service through correspondence; letters requesting information will receive special attention. The Library is prepared to furnish references and copies of articles on mining and metallurgical subjects; to determine the existence of mining maps, and to furnish general information as to the geology and mineral resources of all countries.

All communications should be made as definite as possible so that the information received may be what is desired and not include collateral matter which may not be of interest. The time spent in searching for such collateral matter will be saved, and the information will be sent more promptly and in more usable shape.

The members of the Institute can be of service to the Library by forwarding copies of mining reports; maps privately issued, and similar material, which will be classified, indexed, and made available to other members. Suggestions for additions to the Library, either by purchase or personal solicitation as gifts, will be welcomed. It is hoped that members while in the city will use the Library freely, and assurance is given that most careful service will be rendered to them.

LIBRARY ACCESSIONS**PARTIAL LIST CLASSIFIED BY SUBJECTS****Mining and Metallurgy**

A B C OF IRON AND STEEL. By A. O. Backert. Cleveland, 1915.

BUYING AND SELLING OF ORES AND METALLURGICAL PRODUCTS. (Technical Paper 83, U. S. Bureau of Mines.) Washington, 1915.

CASE HARDENING OF STEEL. By Harry Brearley. London, 1914.

CUYUNA ORES OF MINNESOTA, PRELIMINARY CONCENTRATION TESTS ON. (Bull. 3, Minnesota School of Mines Experiment Station.) Minneapolis, 1915.

EXPLOSIBILITY OF COAL DUST. (Bull. 20, U. S. Bureau of Mines.) Washington, 1911.

NOTES ON MINERS' CARBIDE LAMPS. (Miners' Circular 18, U. S. Bureau of Mines.) Washington, 1915.

RULES AND REGULATIONS FOR METAL MINES. (Bull. 75, U. S. Bureau of Mines.) Washington, 1915.

- RULES FOR HANDLING, STORING, DELIVERING AND SHIPPING EXPLOSIVES. (Pamphlet No. 5, Institute of Makers of Explosives.) 1914. (Gift of Institute of Makers of Explosives.)
- THEORY AND PRACTICE OF ORE DRESSING. - By E. S. Wiard. New York, 1915.
- UNITED STATES MINING STATUTES ANNOTATED. (Bull. 94, pt. 1-11, U. S. Bureau of Mines.) Washington, 1915.

Geology and Mineral Resources

- ALASKA. GEOLOGY AND MINERAL RESOURCES OF KENAI PENINSULA, ALASKA. (Bull. 537, U. S. Geological Survey.) Washington, 1915.
- AUSTRALIA. GEOLOGICAL SURVEY OF CARGO GOLD-FIELD, NEW SOUTH WALES. (Mineral Resources, No. 19.) Sydney, 1915.
- CANADA. GEOLOGY OF THE VICTORIA AND SAANICH MAP AREAS, VANCOUVER ISLAND, B. C. (Memoir 36, Canada Mines Department.) Ottawa, 1913.
- CANADA. SALT DEPOSITS OF CANADA AND THE SALT INDUSTRY. Ottawa, 1915.
- CANADA. GEOLOGICAL RECONNAISSANCE BETWEEN GOLDEN AND KAMLOOPS, B. C., ALONG THE CANADIAN PACIFIC RAILWAY. (Memoir 68, Canada Mines Department.) Ottawa, 1915.
- MEXICO. ECONOMIC GEOLOGY OF THE CANANEA MOUNTAINS, THE CANANEA CONSOLIDATED COPPER CO., S. A., 1908, REPORT. By S. F. Emmons. (Gift of L. D. Ricketts.)
- TASMANIA. ZINC-LEAD SULPHIDE DEPOSITS OF THE READ ROSEBERRY DISTRICT, WITH MAPS. (Bull. 19, Tasmania Geological Survey.) Tasmania, 1915. (Gift of Geological Survey of Tasmania.)
- UNITED STATES. GUIDE BOOK OF THE WESTERN UNITED STATES. Part C, THE SANTE FE ROUTE. (Bull. 613, U. S. Geological Survey.) Part D, THE SHASTA ROUTE AND COAST LINE. (Bull. 614, U. S. Geological Survey.) Washington, 1915.
- MONAZITE, THORIUM, AND MESOTHORIUM. (Technical Paper 110, U. S. Bureau of Mines.) Washington, 1915.
- COLORADO. GEOLOGIC MAP OF COLORADO, 1913. TOPOGRAPHIC MAP OF COLORADO, 1913. (Gift of Colorado Geological Survey.)
- COLORADO. BIBLIOGRAPHY OF COLORADO GEOLOGY AND MINING FROM THE EARLIEST EXPLORATIONS TO 1912. (Bull. 7, Colorado Geological Survey.) Denver, 1914.
- COLORADO. COMMON MINERALS AND ROCKS, THEIR OCCURRENCE AND USES. (Bull. 6, Colorado Geological Survey.) Denver, 1913. (Gift of Survey.)
- COLORADO. CLAYS OF EASTERN COLORADO. (Bull. 8, Colorado Geological Survey.) Denver, 1915. (Gift of Survey.)
- NEVADA. GEOLOGY AND MINERAL DEPOSITS OF THE NATIONAL MINING DISTRICT, NEVADA. (Bull. 601, U. S. Geological Survey.) Washington, 1915.
- TEXAS. MINERAL RESOURCES OF TEXAS. (Bulletin of the University of Texas, Scientific Series, No. 29.) Austin, 1914.
- THE MINERAL INDUSTRY, ITS STATISTICS, TECHNOLOGY, AND TRADE DURING 1914. Volume XXIII. Edited by G. A. Roush. McGraw-Hill Book Co., New York, 1915. Price \$10. (Gift of Publishers.)

[NOTE.—As stated by its editor, the object of Mineral Industry is to bring together as complete a record as possible of the year's progress in the mining and metallurgical industries. Production statistics are included, but are used primarily as an indication of progress. The make-up of the volume is similar to that of preceding volumes, the various minerals being taken up in alphabetical order by specialists in the respective subjects. In addition to the regular features, articles are included on The Flotation Concentration of Ores, showing the present status of this method of ore dressing so far as it could be learned from the various companies concerned; The Smoke Problem in the Mineral Industries, being a summary of the literature on the subjects of the treatment of smoke, fume, and dust arising from industrial plants. The different methods developed to solve the problem are briefly described. This volume supplements the preceding 22 volumes, bringing up to date the statistics of production and showing the developments commercially and technically in the mining and metallurgical industries during 1914.—B. A. R.]

General

- COMPOSITION OF THE NATURAL GAS USED IN TWENTY-FIVE CITIES. (Technical Paper 109, U. S. Bureau of Mines.) Washington, 1915.

- HUMAN EFFICIENCY.** A psychological study of modern problems. By H. W. Dresser. New York-London, 1912.
- DYESTUFFS AND COAL TAR PRODUCTS; THEIR CHEMISTRY, MANUFACTURE AND APPLICATION.** By Thomas Beacall and others. London, 1915.
- PURCHASING.** By C. S. Rindsfoos. New York, 1915.
- ROAD CONSTRUCTION AND MAINTENANCE.** Wilmington, 1915. (Gift of E. I. du Pont de Nemours Powder Co.)
- MINING WORLD INDEX OF CURRENT LITERATURE.** Volume XII, January-June, 1915. By George E. Sisley. Mining World Co., Chicago, 1915. Price \$2. (Gift of Publishers.)

[NOTE.—This volume, as previous volumes, contains a classified list of the articles on mining, engineering, metallurgy, mining geology, mineralogy, etc., which have been published during the first half of 1915. Following each important article is a brief statement giving a general idea of its contents. A system of cross-indexing is used to enable the ready finding of what is wanted on any mining or affiliated subject.—B. A. R.]

Company Reports

- BROKEN HILL PROPRIETARY CO., LTD.** Reports and Statements of Account for Half Year Ending May 31, 1915. Australia, 1915. (Gift of W. E. Wainwright.)
- BROKEN HILL SOUTH SILVER MINING CO.** Reports, Statements of Accounts, etc., for Half Year Ended June 30, 1915. Melbourne, 1915. (Gift of W. E. Wainwright.)

Trade Catalogues

- BESSEMER GAS ENGINE CO.,** Grove City, Pa. The Bessemer Monthly. September, 1915.
- BRAUN CORPORATION,** Los Angeles, Cal.
Braun Iron Ore Grinder, No. 7.
Braun Laboratory Appliances.
- CONTINUOUS WIRE WOUND PIPE CO.,** San Francisco, Cal. Continuous wire-wound wood-stave pipe. 1915.
- DE LA VERGNE MACHINE CO.,** New York, N. Y. Bull. 142. De La Vergne oil engine, type "D H."
- DENVER QUARTZ MILL & CRUSHER CO.,** Denver, Colo. Catalogue No. 12. Denver Quartz Mill.
- GOHEEN MFG. CO.,** Canton, O. Engineers' handbook on protective coatings.
- HARRISON SAFETY BOILER WORKS,** Philadelphia, Pa. Description of Cochran metering heater.
- LESCHEN, A. & SONS ROPE CO.,** St. Louis, Mo. Leschen's Hercules. September, 1915.
- SULLIVAN MACHINERY CO.,** Chicago, Ill. Bull. 72. Sullivan drill sharpener. August, 1915.

MEMBERSHIP

NEW MEMBERS

The following list comprises the names of those persons who became members during the period Sept. 10 to Oct. 10, 1915:

Members

BENEDICT, C. HARRY, Met.....Calumet & Hecla Mining Co., Calumet, Mich.
 BETTERTON, JESSE OATMAN, Met.; Supt. Blast Furnaces and Sulphate Depts.,
 St. Louis Smelt. & Ref. Co., Collinsville, Ill.
 CLARK, H. A., Supt..... Calumet & Arizona Mining Co., Douglas, Ariz.
 COOPER, CLAUDE H., Asst. Supt..... Calumet & Hecla Smelt. Works, Hubbell, Mich.
 CRAMPTON, FRANK A., Min. Engr.; Mine Supt., Platinum Gold Min. Co.,
 Goodsprings, Nev.
 CUNNINGHAM, GEORGE HAMILTON, Asst. Supt. of Construction,
 Anaconda Copper Mining Co., Anaconda, Mont.
 ETTLINGER, ISADOR AARON, Min. Engr..... Magma Copper Co., Superior, Ariz.
 FELLENCER, CHARLES ALLEN, Min. Engr..... 113 So. Madison St., Allentown, Pa.
 HACKFORD, JOHN EDWARD, Chemist..... Aguila Co., Tampico, Mexico.
 HARRIS, GEORGE HENRY RONALD, Min. Engr., 481 Ridout St., London, Ont., Canada.
 HOOD, BEN BENIGHT, Supt. of Boiler Room, U. S. Metals Refining Co., Chrome, N. J.
 HOWLAND, HENRY PHELPS, Supt. of Blast Furnaces, Wisconsin Steel Co., Chicago, Ill.
 HOYT, SAMUEL LESLIE, Metallographer, Asst. Prof. of Metallography,
 University of Minnesota, Minneapolis, Minn.
 HUANG, CHIN TAO, Asst. Chief Met. Engr., Hanyang Iron & Steel Works,
 Hanyang, China.
 IGAWA, TAKEO, Min. Engr..... Ray Consolidated Copper Co., Ray, Ariz.
 JAHN, WILLIAM F., Min. and Met. Engr., Mill Supt., New York & Honduras
 Rosario Min. Co., San Juancito, Honduras.
 JAY, CLYDE HOMER, Vice-Prest. and Mine Mgr., Giant-Eclipse Con. Mines Co.,
 Salida, Colo.
 KOPE, GODFREY SAMUEL, Min. Engr.; Surveyor, May Cons. G. M. Co.,
 Germiston, Johannesburg, Transvaal.
 OLMSTEAD, JOHN STANLEY, Min. Engr.; Asst. Engr., Experimental Leaching,
 New Cornelia Copper Co., Ajo, Ariz.
 OREM, ARCHIE J..... Genl. Supt., Nevada Douglas Copper Co., Ludwig, Nev.
 PERRY, HOWARD E., Mining Operator; Pres. & Mgr., Chisos Mining Co.,
 Terlingua, Tex.
 POTTER, JAMES ALBERT.... Construction Engr., New Cornelia Copper Co., Ajo, Ariz.
 RANDALL, JOHN, Met.; Mill Supt., National Mining & Development Co.,
 Placerville, Idaho.
 ROCKEFELLER, GEORGE EGBERT, Supt..... Fairfax Mine, Inc., Fairfax, Wash.
 SANDS, WILLIAM ELMER, Assayer..... Ophir, Colo.
 SCHETTLER, BRUNO..... Genl. Mgr. of Mine, Benton Coal Co., Benton, Ill.
 SCHNEPP, CHARLES F., Min. Engr..... Ridgefield Park, N. J.
 SHEPARD, FRANK E., Mech. Engr.... Ore Dressing, 924 Washington St., Denver, Colo.
 TISSOT, ARMAND, General Representative..... Ingersoll-Rand Co., Paris, France.
 TROUTMAN, GREYSON PREVOST, Asst. Genl. Mgr..... G. B. Markle Co., Jeddo, Pa.
 WILSON, ELIHU CLEMENT, Prest.... Wilson & Willard Mfg. Co., Los Angeles, Cal.
 WILSON, HERBERT MICHAEL, Director, Dept. of Inspection & Safety,
 The Associated Companies, First National Bank Bldg., Pittsburgh, Pa.
 WOLF, WILLIAM JAMES, Resident Mgr., Four States Coal & Coke Co.,
 Worthington, W. Va.
 WRIGHT, HARRY F. Geol..... Gypsy Oil Co., Tulsa, Okla.
 WYSOR, RUFUS JOHNSTON, Chief Chem. and Engr. of Tests, Bethlehem Steel Co.,
 So. Bethlehem, Pa.

Associate Members

GRACE, WILLIAM RUSSELL..... 11 Broadway, New York, N. Y.
 RAE, ROBERT, Genl. Auditor..... Phelps, Dodge & Co., Inc., Douglas, Ariz.
 STRANEY, JOHN W., Civil Engr..... Braden Copper Co., Rancagua, Chile.
 WATSON, CHARLES REA, Mine Mgr., Cusi Mining Co., Cusiuhirachic, Chih., Mex.

Junior Members

APPLEGATE, JOHN SAM., Student.....1523 Millicent Ave., Youngstown, O.
 BURROUGHS, AMBROSE HAMMET, JR., Mine Engr., Britannia Min. & Smelt. Co.,
 Britannia Beach, B. C., Canada.
 CARSTENS, CARL EBERHARD, Student.....120 Mt. Auburn St., Cambridge, Mass.
 HEAD, JAMES LAWRENCE, Student.....Missouri School of Mines, Rolla, Mo.
 JOHNSON, GUY ROCHE, JR., Student....510 Delaware Ave., So. Bethlehem, Pa.
 MEINZER, GOTTHOLD H., Student.....120 Mt. Auburn St., Cambridge, Mass.
 WOLCOTT, HERBERT BETTS, Mine Owner and Operator, 928 S. Clinton St.,
 Carthage, Mo.
 YEN, CHUANG, Student.....429 Atwood St., Pittsburgh, Pa.
 ZACHARIAS, GEORGE W., Student.....Delta Upsilon House, Stanford Univ., Cal.

CANDIDATES FOR MEMBERSHIP

APPLICATION FOR MEMBERSHIP.—The Institute desires to extend its privileges to every person to whom it can be of service. On the other hand, it is not desirable that persons should be admitted to membership in classes for which they are not qualified. Members of the Institute can be of great service if they will make a practice of glancing through the list of applicants and promptly notifying the Committee on Membership, or the Secretary of the Institute, of any persons whom they think should not be classified in accordance with the list given.

The following persons have been proposed during the period Sept. 10 to Oct. 10, 1915, for election as members of the Institute. Their names are published for the information of Members and Associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Board of Directors, which has the power of final election.

Members

Ivan Harry Barkdoll, Globe, Ariz.

Proposed by P. G. Beckett, L. O. Howard, B. Britton Gottsberger.

Born 1876, Gallatin, Mo. 1901–02, Winston. 1905, C. P. Wingfield, Etta Mine.
 1906, C. P. Wingfield, Lone Star Mine. 1907–13, Copper Queen Cons. Min. Co.

Present position: 1913 to date, Old Dominion Copper Min. & Smelt. Co.

Edgar Earl Barker, Chuquicamata, Chile.

Proposed by D. L. H. Forbes, Frederick Hellman, James S. Wroth.

Born 1884, Grass Valley, Cal. 1906, Univ. of California; B. S. 1906, Assayer, Progreso Mines Co., Triunfo, Baja California, Mexico. 1907–09, Min. Engr., Globe Cons. Copper Co. and Live Oak Dev. Co., Globe, Ariz. 1910–12, Chief Min. Engr., Nevada Cons. Copper Co., Ely, Nev.

Present position: 1913 to date, Mine Supt., Chile Exploration Co.

George L. Clark, Schumacher, Ont., Canada.

Proposed by C. H. Poirier, Maurice W. Summerhayes, Frank C. Loring.

Born 1888, Council Bluffs, Iowa. 1905, Bessemer, Mich., High School. 1906–07, Armour Institute, Chicago, Ill. 1907–10, Michigan College of Mines; E. M. 1903–04 (summer), Assayer's helper; 1905–06, Electrician's helper, Colby mine, Bessemer. 1908–10 (summer), Practical underground work, C. & H., Calumet; Colby mine, Bessemer; Baltic iron mine, Iron River. 1910–12, From time-keeper and cost clerk to shift boss, Creighton Mine, Ont., Can. Copper Co. 1912–13, Supt., Dill Quartz Quarry, Can. Copper Co. 1913, Supt., No. 2 Mine, Can. Copper Co. 1914, Sampler, Engineer's helper, and Asst. Assayer, Hollinger Mine.

Present position: Min. Engr., Vipond Mine.

Richard A. Conkling, Tulsa, Okla.

Proposed by Arthur Thacher, C. L. Severy, H. A. Wheeler.

Born 1885, Bowie, Tex. 1911, Univ. of Oklahoma; A. B. 1911-14, Grad. work in Geology, Univ. of Chicago. 1910 (summer), Aid on U. S. G. S., N. M. 1911 (summer), Aid on U. S. G. S., Alaska. 1912-13, Geological work in Venezuela, Bermudez Co., and Caribbean Petroleum Co. 1914, Royal Dutch Oil Co., The Hague, Holland.

Present position: Head Geol., Roxana Petroleum Co.

William Bronson Cramer, Globe, Ariz.

Proposed by P. G. Beckett, L. O. Howard, Guy N. Bjorge.

Born 1880, Ansonia, Conn. 1899-1902, Sheffield Scientific School, Yale Univ.; Ph. B. 1902-05, Graduate Student and assistant in instruction at Sheffield Scientific School, Yale Univ. 1905-07, Asst. Chem., reduction works, Copper Queen Cons. Min. Co., Douglas, Ariz. 1907-08, Chem., reduction works, Shannon Copper Co., Clifton, Ariz.

Present position: 1909 to date, Met. at concentrator, Old Dominion Copper Min. & Smelt. Co.

Hubert Ingersoll Ellis, Kellogg, Idaho.

Proposed by Milnor Roberts, Joseph Daniels, Henry Landes.

Born 1889, Menlo, Wash. 1897-1905, Menlo grade and high schools. 1905-10; 1914-15, Univ. of Washington, College of Mines; B. S.; 1909-10, Student, Asst. in Met. 1910, Placer prospecting, Fairbanks district, Alaska. 1911, Lessee, Newsboy Ext. quartz claim, Fairbanks. 1912, Miner and millman, Rhoads-Hall mine. 1913, Point-man, Keys & Rettig. 1914, Lessee, Zero claim.

Present position: Underground and in experimental plant, Bunker Hill & Sullivan Min. & Cons. Co.

Henry Groos, Harlan, Ky.

Proposed by Will Ward Duffield, H. M. Waite, Thomas H. Claggett.

Born 1872, Siegen, Germany. 1883-92, Real-Gymnasium, Siegen (Degree for Matriculation). 1892-93, Royal Min. Academy & Geol. Institute, Berlin, Germany. 1893-95, Royal Min. Academy, Clausthal, Germany. 1895-1905, Engaged in Engrg., mine development work with various coal and iron ore companies of Germany and Luxemburg. 1905-10, In charge of triangulations with Clinch Coal Corporation, Dante, Pa. 1911-13, Chief Engr., La Follette Coal, Iron & Ry. Co., La Follette, Tenn.

Present position: 1913 to date, Min. and Cons. Engr., Harlan, Ky.

Charles A. Hansen, Schenectady, N. Y.

Proposed by A. G. MacGregor, Henry A. Tobelmann, David B. Rushmore.

Born 1882, Racine, Wis. 1905, Univ. of Wisconsin; B. S. 1905-08, Research Asst., General Electric Co., Schenectady, N. Y. 1914, Development of wet extraction

Present position: Met., General Electric Co.

Henry Hanson, San Francisco, Cal.

Proposed by G. M. Colvocoresses, Thomas T. Read, Edmund Jüssen.

Born 1874, Blair, Neb. 1894, High School, Deadwood, S. D. 1895-98, Academy, Northwestern Univ., Evanston, Ill. 1898-1902, College of Liberal Arts, Northwestern Univ.; Ph. B. 1892-95, Solution man and night foreman, Portland Cons. Chlorination Plant, Deadwood, S. D. 1895-1902, Worked in various capacities during summer vacation, Homestake Min. Co., Lead, S. D. 1902-03, Homestake Mills, assay office, refinery, worked in various capacities for experience. 1903-07, Foreman, Homestake Cyanide Plant, Lead, S. D. 1907-08, Field Engr., C. W. Merrill; installed cyanide equipment, P. S. G. M. Co., Blair, Nev. 1908-10, Mill Supt., Pittsburgh Silver Peak G. M. Co., Blair, Nev. 1910-12, Field Engr., Merrill Metallurgical Co., in charge of erection of Dome mill with title Supt. of Construction. 1912-13, Mill Supt., Dome mill, So. Porcupine, Ont., Canada. 1913-14) Engineering Staff, Merrill Metallurgical Co., San Francisco, Cal.

Present position: Mill Supt., Engels Copper Co.

Thomas Woodburn Hardy, Jr., Ottawa, Ont., Canada.

Proposed by Stanley N. Graham, Albert Sauveur, H. M. Boylston.

Born 1886, Dublin, Ireland. General education in public and private schools, Nova Scotia. 1904-08, Matriculated into Dalhousie Univ., Halifax, N. S. 1910, Grad., Nova Scotia Technical College; B. S. in Min. Engrg. 1910-11, Draftsman, Nova Scotia Steel & Coal Co., Sydney Mines, N. S. 1911, Chemist and Draftsman, Canada Iron Corp., Bathurst, N. B. 1911-14, Chemist and Metallographist, Nova Scotia Steel & Coal Co., New Glasgow, N. S.

Present position: 1914 to date Asst. Engrg. Chemist, Division of Fuels and Fuel Testing, Dept. of Mines, Canada.

Walter Harris, Globe, Ariz.

Proposed by P. G. Beckett, L. O. Howard, Guy N. Bjorge.

Born 1872, London, England. To 1890, Private tuition. 1890-92, King's College, Cambridge. 1893-94, Camborne School of Mines, Cornwall, England. 1895-96, Student of practical mining and metallurgy, Ophir mine, Grass Valley, Cal. 1903-04, Camborne School of Mines, special course in Geology, Mineralogy and Mine Surveying. 1897-98, Chemistry and Assaying, Davey laboratory, Redruth, Cornwall, England. 1898-1902, Wondalli (Deccan) Gold Min. Co., Ltd., India. 1905-06, Reduction Works, Abosso Gold Mines, Gold Coast, W. Africa. 1907-08, Arizona Commercial Min. Co., Globe, Ariz.

Present position: Chem., Old Dominion Min. & Smelt. Co.

Ray E. Hoffman, Hannibal, Mo.

Proposed by Fred Thomas Agthe, C. R. Forbes, H. A. Buehler.

Born 1881, Knoxville, Iowa. 1905, B. S.; 1908, E. M., Missouri School of Mines. 1905-06, Shannon Copper Co., Clifton, Ariz.

Present position: 1906 to date, Plant Mgr., Asst. Genl. Supt., Atlas Portland Cement Co.

Lemuel Ray Jenkins, Mammoth, Shasta Co., Cal.

Proposed by G. W. Metcalfe, A. P. Anderson, J. H. Kervin.

Born 1882, Revere, Mass. 1901-03, Phillips Exeter Academy. 1903-07, Harvard Univ.; B. S. 1907, Miner, Mammoth Copper Min. Co., Mammoth, Cal. 1908-12, Engrg. Dept., same company. 1912-14, Supt., York Harbor Mine, Newfoundland, for U. S. Exploration Co.

Present position: 1914 to date, Asst. Supt., Mammoth Copper Min. Co.

John M. Lovejoy, Peking, China.

Proposed by J. F. Kemp, William Campbell, Charles P. Berkey.

Born 1889, New York, N. Y. 1912, Columbia University, New York; E. M. 1912, General engineering, Chile Exploration Co., Chuquicamata, Chile. 1912-14, Supt., silver mines, Compania Mineral Arrendataria de Huantajaya, Huantajaya, Chile. 1914, Examination work for F. G. Clarke in South America and H. P. Lowe in Colorado.

Present position: Topographical and geological work in connection with petroleum.

John Chambers Low, Globe, Ariz.

Proposed by P. G. Beckett, L. O. Howard, B. Britton Gottsberger.

Born 1886, Brooklyn, N. Y. 1893-1904, Public schools, Brooklyn, N. Y. 1904-07, Dickinson College. 1907-08, Mine surveying and construction, Lehigh Coal & Navigation Co., Lansford, Pa. 1909, Topographer and transitman, Gila Valley, Globe & Northern R. R., Pima, Ariz., and Miami, Ariz. 1910, Prospecting, assessment work, Globe mining district; churn drill and mine sampling, Inspiration Copper Co., Miami, Ariz. 1910-13, Engrg. Dept., Old Dominion Copper Min. & Smelt. Co., Globe, Ariz. 1914, Sampling, special work in tramming and development, Old Dominion Copper Min. & Smelt. Co.

Present position Sampling, Shift Boss, Old Dominion Copper Min. & Smelt. Co.

Clark G. Mitchell, Victor, Colo.

Proposed by F. K. Brunton, Louis D. Huntoon, Horace F. Lunt, D. W. Brunton,

John W. Finch, John T. Hawkins.

Born 1887, Leadville, Colo. 1909, Yale Univ.; A. B. 1911, Columbia Univ.; E. M. 1911-12, Engr., Britannia Min. & Smelt. Co., Britannia Beach, B. C., Canada.

Present position: 1913 to date, Mgr., Isabella Mines Co.

Harold Lee Norton, Globe, Ariz.

Proposed by P. G. Beckett, L. O. Howard, G. N. Bjorge.

Born 1882, Boston, Mass. 1903, Mass. Inst. of Tech.; S. B.

Present position: Engr., Old Dominion Copper Min. & Smelt. Co.

Howard Coon Parmelee, Denver, Colo.

Proposed by Lawrence Addicks, Joseph W. Richards, Bradley Stoughton.

Born, 1874, Omaha, Neb. 1897, B. S. 1899, A. M., Univ. of Nebraska. 1898, Asst. Chem., Union Pacific R. R. Co., Omaha, Neb. 1900-02, Chief Chem., Globe plant, A. S. & R. Co., Denver, Colo. 1903-05, Cons. Chem., Denver, Colo. 1905-07, Editor, *Mining Reporter*, Denver. 1909, Editor, *Western Chemist and Metallurgist*, Denver.

Present position: 1910 to date, Western Editor, *Metallurgical and Chemical Engineering*.

Louis Ravicz, Minneapolis, Minn.

Proposed by Charles P. Berkey, Peter Christianson, W. H. Emmons.

Born 1891, Minneapolis, Minn. 1910-14, Minnesota School of Mines; E. M. 1914-1915, Dept. of Geology, Univ. of Minnesota; A. M. 1911 (summer), Assayer and Chemist, Banner M. & M. Co., Silver City, Ida. 1912 (summer), Timberman, Republic Iron & Steel Co., Gilbert, Minn. 1913 (summer), Miner, Miami Copper Co., Miami, Ariz. 1914 (summer), Geologic work in British Columbia and Alberta, Ralph Arnold and W. R. Calvert. 1915 (summer), Geologic work in Cuyana iron district, Minn.; E. C. Harder and A. W. Johnston.

Present position: Graduate work, Columbia Univ.

William A. Simkins, Reno, Nev.

Proposed by William Hague, J. B. Stapler, W. L. Fraser.

Born 1883, Coldwater, Mich. 1903-05, Univ. of Mich. (Engrg.). 1905, Miner in Butte district. 1905-07, Surveying, Keystone-Nevada Gold Min. Co., Olinghouse, Nev. 1907-09, Engrg., surveying, Haff & Colwell Bros., Ely, Nev. 1909-10, Engr., Nevada New Mines Co., Rawhide, Nev. 1910-11, General engineering, Nevada and California.

Present position: 1911 to date, Lloyd & Sinkins.

Tom Simpson, Johannesburg, Transvaal, So. Africa.

Proposed by A. J. Brett, R. C. Warriner, Palmer Carter.

Born 1876, London, England. 1894-98, Associateship of the Royal School of Mines, London, England, in Mining. 1898-99, Assayer and sampler, Gold Coast, W. Africa. 1899-1900, Asst. Mgr., lead mines, N. Wales. 1903-04, Sampler, Robinson G. M. Co., Johannesburg. 1904-05, Shift boss, French Rand G. M. Co., Johannesburg. 1905-07, Shift boss, Ferreira G. M. Co., Johannesburg. 1907-09, Mine Captain, E. R. P. M., Johannesburg. 1909, Mine Captain, Robinson Central Deep G. M. Co., Johannesburg. 1909-12, Underground Mgr., Crown Mines, Johannesburg.

Present position: 1912 to date, Joint Mgr., Crown Mines.

Walter Crispell Smith, Chrome, N. J.

Proposed by Sidney Rolle, Francis R. Pyne, R. W. Deacon.

Born 1884, Kingston, N. Y. 1901-03, Brooklyn Polytechnic Institute. 1903-06, Lehigh Univ.; A. C. 1906-08, Asst. Research Chemist, Chrome Plant. 1908-09, Foreman; 1909-14, Met., United States Metals Refining Co., Grasselli, Ind., plant.

Present position: 1914 to date, Supt., United States Metals Refining Co.

Philip D. Wilson, Warren, Ariz.

Proposed by Ira B. Joralemon, Arthur Notman, Gerald Sherman.

Born 1888, Chicago, Ill. 1905-09, Princeton Univ.; B. S. 1909-11, Columbia School of Mines; E. M. 1910, Mining, Bingham New Haven Copper & Gold Min. Co., Bingham, Utah. 1911, Chemist, Copper Queen Reduction Works, Douglas, Ariz. 1912, Engr., Transvaal Copper Min. Co.; Cumpas, Son., Mexico. 1913, Geol., Shattuck Arizona Copper Co., Bisbee, Ariz.

Present position: 1914 to date, Geol., Calumet & Arizona Min. Co.

*Associate Members***Algernon Sidney Uhler, Johannesburg, So. Africa.**

Proposed by W. L. Saunders, George A. Howells, George T. Cousins.

Born 1874, Philadelphia, Pa. Various preparatory schools, Philadelphia, Pa. 1890, Georgetown College, Washington, D. C. 1896, Univ. of Pennsylvania, Mech. Engrg.

Present position: 1897 to date, Mgr., Ingersoll-Rand Co.

Howard Thornton Walsh, Chicago, Ill.

Proposed by J. A. Caselton, E. J. Rossbach, F. K. Copeland.

Born 1878, Newburgh, N. Y. 1886-94, Public schools, New York and Massachusetts. 1894-96, Private school. 1896-1900, Worcester Polytechnic Institute, Worcester, Mass.; B. S. Eleven years in West, two years in Europe, two years in Chicago.

Present position: 1900 to date, Sales Mgr., Sullivan Mach. Co.

Junior Members

Guy C. Faust, State College, Pa.

Proposed by W. R. Crane, W. M. Weigel, C. E. McQuigg.

Born 1889, Pottsville, Pa. 1908, Grad., Pottsville High School. 1910, Grad., Franklin and Marshall Academy, Lancaster, Pa. 1908-09, 1910-12, Engrg. Corps, Philadelphia & Reading Coal & Iron Co. 1914-15, P. W. Sheaffer Estate, Pottsville, Pa.

Present position: Student, School of Mines, Pennsylvania State College.

Clarence Korer Hall, State College, Pa.

Proposed by C. E. McQuigg, W. R. Crane, E. S. Moore.

Born 1893, Harrisburg, Pa. 1912, Harrisburg High School.

Present position: Student, Pennsylvania State College.

John Campbell Herr, State College, Pa.

Proposed by C. E. McQuigg, W. R. Crane, E. S. Moore.

Born 1893, Butler, Pa. 1911, Butler High School, Butler, Pa. 1911 (summer), Gauger in rolling mill, Forged Steel Wheel Co., Butler, Pa. 1915 (summer), Spring fitter or maker, Standard Steel Car Co., Butler, Pa.

Present position: Student, Pennsylvania State College.

Reginald Phillipps Oliveros, Savannah, Ga.

Proposed by William R. Chedsey, Harry J. Wolf, W. G. Haldane.

Born 1891, Savannah, Ga. 1905-07, Savannah High School. 1907-08, Georgia Tech. 1908-10, Preparation for Navy. 1910-11, Alabama Polytechnic Institute. 1911-15, Colorado School of Mines. 1914, Asst. Chem., American Agriculture Chemical Co., Savannah, Ga.

Present position: June, 1915, to date, Acting Asst. Engr., Liberty Bell Gold Mining Co., Telluride, Colo.

Herbert Henry Shaver, State College, Pa.

Proposed by W. R. Crane, W. M. Weigel, C. E. McQuigg.

Born 1894, Wilkes-Barre, Pa. 1912, Winfield Min. Co., Webb City, Mo. 1914, Lehigh Valley Coal Co., Hazelton, Pa. 1914, Lehigh Valley Coal Co., Wilkes-Barre, Pa.

Present position: Student, School of Mines, Pennsylvania State College.

Roger Lytle Wensley, New York, N. Y.

Proposed by Robert Peele, E. J. Hall, H. L. Carr.

Born 1894, Brooklyn, N. Y. 1909-13, Boys' High School, Brooklyn, N. Y. 1913 (summer), Besson & Co., Dobbs Ferry, N. Y.

Present position: Student, Columbia School of Mines.

Fred P. Williams, New York, N. Y.

Proposed by E. J. Hall, H. L. Carr, Robert Peele.

Born 1886, Riverside, Iowa. 1900-04, High School, Larimore, N. D. 1905-10, Univ. of N. D., Grand Forks, N. D.; M. E. 1906-08, summers in Coeur D'Alenes, British Columbia and Montana, on placers, and on underground work, Hunter Min. Co. 1909, Twin Peaks Min. Co., Duncan, Ariz. 1910, Student. 1911, Whitman Min. Co., Pearl, Idaho. 1912, Utah Copper Co., Bingham, Utah. 1913-14, Chino Copper Co., Santa Rita, N. M.

Present position: Student, Columbia School of Mines.

CHANGES OF ADDRESS OF MEMBERS

The following changes of address of members have been received at the Secretary's office during the period Sept. 10 to Oct. 10, 1915. This list, together with the list published in *Bulletin* Nos. 100 to 106, April to October, 1915, and the foregoing list of new members, therefore, supplements the annual list of members corrected to Mar. 1, 1915, and brings it up to the date of Oct. 10, 1915.

ALSDORF, FREDERICK C. Wickenburg, Ariz.
ANDERSON, AMIL A., Met. Homestake Min. Co., Lead, S. D.
ANDERSON, FRANK M., Geol. Standard Oil Co., 2604 Etna St., Berkeley, Cal.

- ANDERSON, JAMES S. Instructed to hold all mail.
 AUSTIN, JOHN F. Farmington, Conn.
 BATEMAN, ALAN M., Min. Geol. Drawer C, Yale Station, New Haven, Conn.
 BEESON, ALEXANDER C. Worthington, W. Va.
 BENNETT, ARTHUR F. Salmon, Idaho.
 BERRYHILL, JAMES G. 2737 Claremont Blvd., Berkeley, Cal.
 BISSELL, ROBERT W. 611 Arrott Bldg., Pittsburgh, Pa.
 BLAYLOCK, D. W. Bonne Terre, Mo.
 BLEECK, A. W. G. 37 Brook Green, London, W., England.
 BONSAK, ARTHUR A. 1010 Boatmen's Bank Bldg., St. Louis, Mo.
 BRADT, MAURICE L. Theta Tau House, Houghton, Mich.
 BROOKE, LIONEL, Care Chile Exploration Co., Chuquicamata, via Antofagasta, Chile.
 BURLINGAME, WALTER E. Assayer, Chem., 17 6 Lawrence St., Denver, Colo.
 CAHEN, JAMES P., JR. 431 Chestnut St., S. Bethlehem, Pa.
 CALLAWAY, L. A. 613 Walnut St., Anaconda, Mont.
 CARPENTER, CLARK B., Prof. of Chemistry and Metallurgy, State School of Mines,
 Weir City, Kan.
 CARR, HOMER L. School of Mines, Columbia Univ., New York, N. Y.
 CARROLL, FRED. Commissioner of Mines, State Capitol, Denver, Colo.
 CLEAVELAND, NEWTON. 433 California St., San Francisco, Cal.
 COX, W. RAY, Prof. of Min. & Met., Oregon Agricultural College,
 Mines Bldg., Corvallis, Ore.
 CUSHING, DANIEL, Met. Sandusky Foundry & Machine Co., Sandusky, O.
 DALEY, STEPHEN H., JR. Box 943, Niagara Falls, N. Y.
 DAVIS, ROY HENRY. 6 Llewellyn Place, Orange, N. J.
 DESHLER, GEORGE O. Care The Darby Mill, Mazuma, Humboldt Co., Nev.
 DICKERMAN, A. L. 814 North Tejon St., Colorado Springs, Colo.
 DICKMAN, R. N. 170 Bay St., St. Augustine, Fla.
 DOWNS, FLETCHER GARDNER. 183 Carolina Ave., Pinners Point, Portsmouth, Va.
 DUNN, THEODORE S., Prof. of Mining, So. Dakota State School of Mines,
 Rapid City, S. D.
 EYOUB, DJEVAD. Livingston Hall, Columbia Univ., New York, N. Y.
 FILTEAU, C. A. Instructed to hold all mail.
 FISCHER, SIEGFRIED. Care Mrs. H. Kircher, 29 W. 93d St., New York, N. Y.
 FLESCCHUTZ, JOHN C. 620 Gas & Electric Bldg., Denver, Colo.
 FORBES, J. HYDE. Care J. P. Lippincott, 1100 Central Bldg., Los Angeles, Cal.
 FOX, WALTER V. Sierra Plata Min. Co., La Presena, Parral, Chih., Mexico.
 GABY, WALTER E., Asst. Geol. Anaconda Copper Mining Co., Butte, Mont.
 GEIGER, ARTHUR W. Instructed to hold all mail.
 GORDON, JOHN. Care Gordon & Rogers, 141 Broadway, New York, N. Y.
 GRIEVE, RICHARD P., Engrg. Dept. P. S. C., Shaft 3, Clark St., Brooklyn, N. Y.
 HALDEMAN, GEORGE T. 182 N. Franklin St., Wilkes-Barre, Pa.
 HAMANN, WILLIAM AUGUST, JR. Box 107, McGill, Nev.
 HARDER, EDMUND C. 206 Science Hall, Madison, Wis.
 HENDRICKS, GEORGE F., Min. Engr., Mining Corporation of Canada,
 Cobalt, Ont., Canada.
 HESS, RUSH M. Care Harold M. Heas, 1215 Amherst Pl., St. Louis, Mo.
 HOFSTRAND, OSCAR B. Flat River, Mo.
 HOLDERER, GEORGE B., Min. Engr. 165 Broadway, New York, N. Y.
 HONNOLD, WILLIAM L. Hyde Park Hotel, Knightsbridge, London, England.
 IMHOFF, WALLACE G., Met. Engr. Corrigan, McKinney & Co., Cleveland, O.
 JUDSON, WILBER. 602 N. Washington Ave., Lansing, Mich.
 KANE, JOHN I., Min. Engr., American Smelt. & Ref. Co., 1112 Mills Bldg.,
 El Paso, Tex.
 KEENEY, ROBERT M. Care Standard Chemical Co., Canonsburg, Pa.
 KELLY, JAMES A. Pocahontas Coal & Coke Co., Seattle, Wash.
 KNOWLES, SILAS A. 400 Carrie St., Sault Ste. Marie, Mich.
 LAMB, ROBERT B. 27 Cedar St., New York, N. Y.
 LONDON, STEPHEN L. Box 321, Tombstone, Ariz.
 LARSON, ANDREW G. Paulsen Bldg., Spokane, Wash.
 LEDYARD, LEWIS WILLIAM, Min. Engr.; Supt., Teck-Hughes Gold Mines, Ltd.,
 Kirkland Lake, Ont., Canada.
 LEVY, ARTHUR G. 1923 Madison Ave., New York, N. Y.
 LOMBARDI, MAURICE E., Supt. of Oil Developments, Kern Trading & Oil Co.,
 San Francisco, Cal.
 LONGYEAR, J. M., JR. 406 N. Pinckney St., Madison, Wis.

LUTHER, E. C.	Rooms 302, 303, Thompson Bldg., Pottsville, Pa.
McCLURE, ALEXANDER P.	Detroit Iron & Steel Co., Detroit, Mich.
McDONALD, THOMAS	Youngstown & Canfield Rd., Box 4, Youngstown, O.
MARSH, ROBERT, JR.	Care Nevada Cons. Copper Co., Ruth, via Cobre, Nev.
MATLACK, E. V.	515 Bompert St., Webster Groves, Mo.
MEANS, ALAN HAY	Silver Bell, via Red Rock, Ariz.
MILLARD, WILLIAM J.	Care Angus Marshall, Box 707, Port Arthur, Tex.
MILLER, HERBERT F., JR.	1445 S. Park Ave., Lackawanna, N. J.
MITCHELL, GEORGE	Middlemarch, Cochise Co., Ariz.
MONTANDON, GEORGE E.	Andes Exploration Co., Chanaral, Chile.
MORROW, JOHN T., Factory Products Export Corp.,	61 Broadway, New York, N. Y.
NEBEKER, A. C., Supt.	Leonora Mine, Milford, Utah.
NEWCOMB, CLIVE S., Mech. and Met. Engr.,	7a Calle Zacatecas, 182, Mexico, D. F.
NISHIO, KANJIRO	10 Koya-machi, Shizuoka, Japan.
PARSONS, FLOYD W.	Editor, <i>Coal Age</i> , 10th Ave. & 36th St., New York, N. Y.
PERRY, HOWARD E., Mining Operator, Suite 5-6, 51½ Exchange St.,	Portland, Me.
PFEIFFER, GEORGE W.	5020 N. 15th St., Philadelphia, Pa.
PITMAN, S. M.	Care Stirling S. Smith, Box 220A, R. F. D 1, San Diego, Cal.
PORTER, JESSE C., Asst. Supt.	Matahambre Mine, Cuba.
PRATT, WALLACE E.	Formoso, Kan.
PRINCE, ERNEST	Boatmen's Bank Bldg., St. Louis, Mo.
PRYER, WILLIAM CRISTY	307 W. 7th St., Anaconda, Mont.
PUTNAM, BENJAMIN R.	Box 25, R. F. D. 1, Exeter, Cal.
RAIBER, NICOLAUS H.	127 Madison St., Wilkes-Barre, Pa.
REGER, DAVID B., Asst. Geol.	W. Va. Geological Survey, Morgantown, W. Va.
ROBBINS, HALLET R.	Box 81, College Station, Pullman, Wash.
ROCKWELL, FRED GORE	Gwinn, Mich.
SAVAGE, LESLIE L.	143 Riverside Drive, New York, N. Y.
SCOLES, JOHN C.	Bessemer, Mich.
SINGMASTER, J. A., Genl. Supt.	New Jersey Zinc Co., Palmerton, Pa.
SKINNER, EDMOND N.	25 Broad St., New York, N. Y.
SLEEMAN, H. R.	Palace Hotel, Perth, W. Australia.
SMALLEY, WILLIAM W.	Instructed to hold all mail.
SMITH, HOWARD C.	Care Ray Cons. Copper Co., Ray, Ariz.
SOHNLEIN, M. G. F., Min. Engr., Mill Mgr., Compania Minera de Oruro,	Machacamarca, Bolivia.
STROUD, B. K.	333 Pacific Ave., Oakland, Cal.
SUYDAM, A. G.	107 Hayden St., Cripple Creek, Colo.
THOMAS, WILLIAM S.	601 East St., Redding, Cal.
THOMPSON, LESTER S.	251 E. Kingsbridge Road, New York, N. Y.
THURSTON, E. C.	Care Mrs. E. L. Griffith, Ross, Marin Co., Cal.
TSAI, HSIANG	424 Hartley Hall, Columbia University, New York, N. Y.
TURNER, THOMAS NORTON	Ruby, Alaska.
VERRILL, CLARENCE S., Cons. Min. Engr., and Met.; B. C. Representative,	Tonopah Belmont Dev. Co., Vancouver, B. C., Canada.
WALKER, MYRON R., Min. Engr.	Cerro de Pasco Min. Co., Cerro de Pasco, Peru.
WEIGALL, HENRY S., Genl. Mgr., Chosen Gold Mines, Ltd.,	Kok Kang Kol Mine, Chungan, Chosen, Korea.
WHITMAN, ALFRED R.	Box 533, Cobalt, Ont., Canada.
WILLIAMS, JOHN T.	122 W. 76th St., New York, N. Y.
WILLIAMS, RALPH B.	81 Gibbs St., Brookline, Mass.
WILLIAMS, ROBERT Y., Safety Engr., Delaware & Hudson Co., Coal Dept.,	Scranton, Pa.
WILMOT, H. CLIFFORD, ...	Care Engineers' Club, 32 W. 40th St., New York, N. Y.
WOLF, W. A.	Casilla 655, Guayaquil, Ecuador.
WOODWORTH, GUY THOMAS, Student, Montana State School of Mines, Butte, Mont.	
WOOLF, WALLACE G.	Care Tintic Milling Co., Silver City, Utah.
WORTH, JOHN G.	Room 1416, 66 Broadway, New York, N. Y.
YOUNG, JAMES S., Min. and Met. Engr.	Care Milne & Co., Lima, Peru.

ADDRESSES OF MEMBERS AND ASSOCIATES WANTED

Name.	Last address of Record, from which Mail has been Returned.
CAMPBELL, W. C.	4 Princess St., Roodeport, Transvaal, S. Africa.
COOK, PAUL RICHARDSON	4159 Grand Boulevard, Chicago, Ill.
GRAVES, McDOWELL	949 Atlantic Ave., Long Beach, Cal.

HAMILTON, ARTHUR L.	701 W. 14th Place, Chicago, Ill.
HELM, J. D.	Apartado 1277, Mexico City, Mex.
HOLDEN, ROY J.	Blacksburg, Va.
KURIE, F. M.	625 I. W. Hellman Bldg., Los Angeles, Cal.
LE NOIR, FRANK H.	Douglas, Alaska.
MILLER, FRANK BARTON	Blair, Esmeralda Co., Nev.
NASH, WILLARD H.	297 Delaware Ave., Buffalo, N. Y.
PREYTMAN, FRANK REMINGTON	Oyster Bay, N. Y.
PRICHARD, WILL A.	Room 711, 111 Broadway, New York, N. Y.
RHODES, W. B.	Golden, Colo.
SCHINDLER, DONALD F.	215 N. Murray St., Madison, Wis.
SEIBERT, PERCY A.	Hagerstown, Md.
SOMERS, RANSOM E.	804 E. Seneca St., Ithaca, N. Y.
STILLWAGON, SAM C.	329 W. 58th St., New York, N. Y.
SULLIVAN, WILLARD P.	Henry Walks Co., Norfolk, Va.
WELSH, NORVAL J. E.	Organ, P. O., N. M.
WILSON, JOHN B.	Cerro de Pasco, Peru.

NECROLOGY

The deaths of the following members were reported to the Secretary's office during the period Sept. 10 to Oct. 10, 1915:

Date of Election.	Name.	Date of Decease.
1913	*Hebgen, Max.	August 24, 1915.
1881	**Torrey, H. G.	August 29, 1915.

* Member.

** Life Member.

BIOGRAPHICAL NOTICES

Max Hebgen, who died at Chicago, Aug. 24, 1915, was one of the industrial leaders of Montana and Idaho. The following account of his life and work, taken substantially from the *Anaconda Standard* of Aug. 25, not only depicts a remarkable career, but also presents a noteworthy review of mechanical progress in the far Northwest.

At the time of his death, Mr. Hebgen was Vice-President and General Manager of the Montana Power Co., which, together with its subsidiary corporations, constitutes, perhaps, the largest organization of its kind in the world; and he had advanced to this high position as a "captain of industry" through successive promotions, earned by patient practice, unswerving loyalty and unceasing study, aided by extraordinary executive ability.

Max Hebgen was born in 1870 at Beaver Dam, Wis., and received an average education in the schools of that place and of Milwaukee. In 1887, he entered the Law School of the University of Wisconsin, but left at the end of the first term, in order to enter the service of the Westinghouse Electric Co. From that time on, his life was devoted to electrical engineering. At the age of 20 he came to Butte, where he resided for a quarter of a century, building up and developing the electrical resources of Montana. He became associated with W. A. Clark, the late P. A. Largey and the late Harry A. d'Acheul as Superintendent of the old Butte Electric Lighting Co. This was succeeded by the Butte General Electric Co., which was merged with the Silver Bow Electrical Co. into the Butte Electric & Power Co. Mr. Hebgen was advanced from Superintendent to General Manager of the last-named company and then to the general managership, a position which then required a person

with business courage as well as ability for organization and superior technical knowledge. When the Montana Power Co. was organized, he became the director of its activities with the late John G. Morony, whose death was a severe blow to Mr. Hebgen. In 1910 he was Superintendent of the Butte Electric & Power Co. and the Great Falls Power Co. and then became Vice-President, General Manager and director of these companies.

Mr. Hebgen had millions at his command. Those who intrusted him with directing the hydro-electrical power development of the State had such confidence in him that it is related that when he once said, "I need three millions for this work," he was told, "You can have it." And his judgment was vindicated. At the time of his death he was in charge of properties developing 225,000 horsepower. One of his first jobs as an engineer was done on the Big Hole River, 22 miles south of Butte. This plant was built in 1898 and carries power to the city of Butte for the street railway company and for the pumping plant of the Butte Water Co. The development was made by building a rock-filled crib dam across the narrow canyon of the river, giving a head of 65 ft. The Canyon Ferry plant on the Missouri, 17 miles from Helena, was built in 1898 and 39 ft. of head are developed there. Power from this plant is used principally to supply the requirements of the mining industry. It was only a few years ago that the Montana Power Co. began to build its steel frames for carrying high-voltage wires to supply the Butte mines and the Anaconda smelter with electricity.

The Madison plant, No. 1, was built in 1901 on the Madison River canyon, 61 miles east of Butte, the development originally consisting of a low crib dam and wooden flume about 1,000 ft. long. The power was transmitted to Butte.

In 1890, under Mr. Hebgen's direction, a dam was built across the Missouri River just above the Black Eagle Falls, near the city of Great Falls, and 8,300 hp. developed, of which 7,500 was used directly at the Boston & Montana smelter in Great Falls and the rest transmitted electrically and used for lighting and street-car purposes. These four plants supplying power for Butte, Helena, and Great Falls were built when the transmission of power by electricity was in its infancy and formed the nucleus of what has become one of the great power systems of the country.

It was after 1905 that great strides were made under Hebgen's direction. The Madison River dam was rebuilt and enlarged and the Madison No. 2 plant was installed. The wood-stave pipe, 7,400 ft. long, was at the time the largest conduit of this type in the world. Additional transmission lines were built to Butte, to Alder Gulch, to Three Forks, and to Bozeman, Livingston, and later Billings.

On the Missouri River below Canyon Ferry was constructed the Hauser Lake plant, which furnished power to Butte and Anaconda and to small mines between Butte and Helena.

Perhaps the greatest monument to the courage of Mr. Hebgen is the Great Falls system. In a distance of 8 miles the river near Great Falls drops 400 ft., half being in abrupt falls, making power development easy and a total of 125,000 kw. feasible. In 1910 the Rainbow dam was built. Power from this plant is transmitted to the smelter at Great Falls and to Butte over duplicate steel tower transmission lines. From Butte he caused the construction of an additional line to Anaconda and feeders were run from the Rainbow plant to Lewistown, Havre, and Cascade. In 1906, a dam was built on Missoula River below the mouth

of the Blackfoot, and from this place Missoula is served and a high-tension line serves the cities of Hamilton, Stevensville, and Victor.

Mr. Hebgen's next activity was the 2,500-hp. development on the Big Fork River, in the northwestern part of the State. Power from this source now serves the cities of Kalispell, Whitefish, Columbia Falls and intermediate points.

Early in August, 1915, water went over the spillway of the Big Falls dam, 20 miles from Great Falls. This dam is the largest of three dams of the company on the Missouri River, and its construction was one of the notable feats in Mr. Hebgen's engineering career.

Another plant which was completed in the last year of Mr. Hebgen's life is that of the Thompson Falls Power Co. on the Clark fork of the Columbia River.

In November, 1914 Mr. Hebgen undertook the work of supplying power and the erection of poles and wires for the Milwaukee electrification. His plans were wisely made, and the power company is ahead of its contract, as the first units of the Thompson Falls plant are running and are ready to supply power for part of the 400 miles of track from Avery, Idaho, to Harlowton, Mont. This work attracted attention to Mr. Hebgen from all sections of the country.

While Mr. Hebgen's activities were largely directed toward the hydro-electrical development of Montana, he was also a leader in the utilization of water sites in Oregon and Idaho. Several years before his death, he became interested in behalf of the Montana Power Co. in developments on Henry's fork of the Snake River, near its head-waters in eastern Idaho, and near the Montana State line. He looked into the possibilities of water-power development at the Ox Bow site of the Idaho-Oregon Water Power & Townsite Co. Mr. Hebgen had also been asked for his opinion on sites in northern Idaho.

The plants in Montana, which had been built and developed by Mr. Hebgen within the last 15 years, can be summarized as follows:

Big Hole, Billings No. 1, Black Eagle, Canyon Ferry, Hauser Lake, Lewistown No. 1, Lewistown No. 2, Livingston, Madison River No. 1, Madison River No. 2, Prospect Creek, Rainbow, Missoula River development, Big Fork development, Rainbow and Thompson Falls.

The technical exactness of Mr. Hebgen was shown by the success of his transmission system. The various lines are tied together through transformers and operated as one large distributing network into which power is fed from 13 plants and from which power is taken to 46 sub-stations. One year ago there were 1,300 miles of transmission line, of which 340 was steel tower line. The system covers a territory approximately 200 miles square, and supplies power and light for 40 cities and towns and has made it possible to pump by electricity water supplies of all large cities.

In the commercial development of the uses of electricity Mr. Hebgen played a large part. A year ago, he pointed with pride to the fact that there were 10,000 electric flatirons in homes supplied with light by his company, where washing machines also and other household apparatus were run by electricity.

Under his direction contracts to supply power for the street-railway systems of Butte, Great Falls, Missoula, Anaconda, and Billings and the Gallatin Valley railway system were executed. The bringing of electric power to the mining districts meant that many mines and mills, operated

at a loss under steam power, were set a-going again at a profit. It was Mr. Hebgen who saw the possibilities of power for mine work, and the immense power possibilities of the Missouri River lay dormant until he decided to put his opinion into effect. A total of 44,000 kw. was used in mines and smelters at the time of Mr. Hebgen's death.

The first railroad to be electrified in the West was the Butte, Anaconda & Pacific. Mr. Hebgen, who had charge of this work, told a conference of railroad men that it was only a matter of time until all transcontinental railways would have recourse to electrical operation, and he was working on broad plans to make this a reality in the intermountain country.

He turned his attention also to supplying power for irrigation projects which involved pumping. The first large plant of this nature he installed in the Prickly Pear Valley near Helena for the Montana Reservoir & Irrigation Co., which supplies water for 5,000 acres. Under his direction the Great Falls Power Co. entered into a contract with the government to furnish power for building dams and canals in connection with the Sun River project. He reported favorably on the proposal to supply water during the irrigation season from a storage reservoir near the headwater of the Madison River close to Yellowstone Park, and the Montana Reservoir & Irrigation Co. is now building a storage reservoir with a capacity of 15,000,000,000 cu. ft.

In anticipation of still greater demands for power Mr. Hebgen had made plans for the development of sites in the northwestern part of the State, especially around Flathead Lake, Kootenai Falls, etc.

Although Mr. Hebgen began in a humble way in Butte, he was gaining international recognition at the time of his death. Many residents remember him as a young man familiarly called "Max" who in workman's garb ran a small plant at what is now the Cabbage Patch, 20 years ago. The steady progress of those 20 years made him the chief factor in that marvelous progress which has supplied to a wide, rich region its one great need—mechanical power. Many poetic souls have rhapsodized about the utilization of water-power as a substitute for coal. Mr. Hebgen turned that vision into a great, continental fact.

Mr. Hebgen was a big man physically, and the ailment which fastened itself on him was more dangerous to a man of his physique. His indomitable energy and determination to push to completion some of the larger projects kept him at his work long after he should have been taking a rest. As a consequence, we have to lament his death from heart disease at the early age of 45.

Herbert Gray Torrey was born in 1841, at New York City, and graduated in 1860 from the College of the City of New York. His father, Dr. John Torrey, was famous chiefly as a botanist, though eminent as a professor of chemistry and geology also, at different times in several American colleges. In 1854, President Pierce appointed Dr. Torrey Chief Assayer in the U. S. Assay Office at New York—a position which he retained until his death in 1873. Herbert Torrey, who had been his father's assistant in that office, became Chief Assayer at his father's death, and continued to serve until 1910, when he went into business for himself as an assayer and consulting metallurgist. He was also a consulting chemist, a government expert in textile fabrics, and an examiner of mines. In the latter capacity he made many trips through the western

United States. He joined this Institute in 1881, and was a frequent attendant at its meetings, together with his wife. He was also a member of the American Society of Mechanical Engineers, the Franklin Institute of Philadelphia, and the Society of the Cincinnati.

In 1912 he retired from active business, residing thereafter at Stirling, N. J. and (in the winter) at San Diego, Cal. In Stirling, where he had lived for many years, he was known as a public-spirited and progressive citizen. He was active in the establishment of the first public school there, the creation of the cemetery, the park gardens, the first volunteer fire company, etc., and served for many years as an elder of the Presbyterian church. He died Aug. 29, 1915, at his home in Stirling, painlessly and suddenly, of heart weakness. Many of the older members of the Institute will sadly miss the genial companionship of Mr. Torrey, and sincerely share the sorrow of his widow.

EXECUTIVE COMMITTEES OF LOCAL SECTIONS

New York

Meets first Wednesday after first Tuesday of each month.

DAVID H. BROWNE, *Chairman*,JOHN H. JANEWAY, *Vice-Chairman*.F. E. PIERCE, *Secretary*, 35 Nassau St., New York, N. Y.P. A. MOSMAN, *Treasurer*.

LEWIS W. FRANCIS.

Boston

Meets first Monday of each winter month.

HENRY L. SMYTH, *Chairman*.ALFRED C. LANE, *Vice-Chairman*.HENRY A. WENTWORTH, *Secretary-Treasurer*, 60 India St., Boston, Mass.

ROBERT H. RICHARDS,

ALBERT SAUVEUR.

Columbia

Holds four sessions during year. Annual meeting in September or October.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close April 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

The Disseminated Copper Ores of Bingham Canyon, Utah

BY J. J. BEESON, STANFORD UNIVERSITY, CAL.

(New York Meeting, February, 1916)

CONTENTS

	PAGE
I. PRIMARY MINERALIZATION.....	2191
1. Introduction.....	2191
a. Scope of Work.....	2191
b. Acknowledgments.....	2192
c. The Bingham Mining District.....	2192
d. General Geology.....	2192
e. The Utah Copper Mine.....	2193
f. Field and Laboratory Work.....	2193
2. The Quartz-Monzonite.....	2212
3. The Pegmatite Veins.....	2213
4. The High-Temperature Veins.....	2214
5. The Early Hydrothermal Period.....	2215
6. The Late Hydrothermal Period.....	2220
II. SECONDARY ENRICHMENT.....	2221
1. Author's Note.....	2221
2. Factors Governing Enrichment.....	2222
3. The Enrichment.....	2222
4. The Covellite Enrichment.....	2223
a. The Capping above the Covellite Ores.....	2223
b. Conclusions Drawn from a Study of the Capping and the Covellite Ores.....	2227
c. Covellite Enrichment at Depth.....	2227
5. The Chalcocite Enrichment.....	2229
a. The Carbonates in the Capping above the Chalcocite Ores.....	2229
b. The Formation of Chalcocite.....	2229
6. Secondary Chalcopyrite and Bornite.....	2230
III. CONCLUSIONS AS TO THE FORMATION OF THE IMPORTANT SULPHIDES IN THE DISSEMINATED ORES.....	2235

I. PRIMARY MINERALIZATION

1. Introduction

a. Scope of Work.—The first part of this paper deals with the country rock, its pegmatitic, pneumatolytic, and hydrothermal alterations; also, the effect of magnetite on the deposition of the primary chalcopyrite and pyrite.

The second part of the paper includes a detailed account of the processes of secondary enrichment which have been effective in producing the disseminated ores. While both parts present new features, it is hoped that the discussion on secondary chalcopyrite and bornite may be of interest to those who are engaged in the study of secondary processes in other districts.

The excellent work¹ *Economic Geology of the Bingham Mining District*, by J. M. Boutwell, was published in 1905. Since that time the extensive development work—surface cuts and underground drifting—of the Utah Copper Co. has made possible the collecting of samples for more detailed microscopic study. The writer has taken advantage of this fact and presents the following work, not as a criticism, but as a continuation of the work which was begun by Mr. Boutwell.

b. Acknowledgments.—The present study was undertaken at the suggestion of Dr. A. F. Rogers; to him, and to Prof. C. F. Tolman and Prof. S. W. Young, the writer wishes to express his thanks for criticism in the preparation of this paper.

The writer also expresses his thanks to D. C. Jackling, Vice-President and Managing Director of the Utah Copper Co., for permission to publish this paper, and to William Spencer, D. W. Wadleigh, T. S. Carnahan, J. A. Caulfield, Percy Dayer, and Edward Norsefall, for the courtesies extended to him while a co-worker in the mines.

c. The Bingham Mining District.—The Bingham mining district is situated in the Oquirrh Mountains, 18 miles southwest of Salt Lake City, Utah. The productive region is limited to an area of about 15 square miles; but the largest producing mines are included in an area of about 2 square miles in the southwestern end of the district.

d. General Geology.—Since the general geology, maps, sections, and detailed descriptions of the different types of ore deposits appear in Mr. Boutwell's report, the writer will only summarize briefly the general geology, emphasizing certain important points mentioned by Mr. Boutwell.

The sedimentary section exposed in this district embraces several thousand feet of massive quartzite, thin interlaced limestones, and calcareous shales of the Upper Carboniferous (Pennsylvanian) age. These sediments have been cut by an intrusive quartz-monzonite which forms connected irregular dikes and masses, in a strip about 1 mile wide and 4 miles long, extending across the southern corner of the district in a north-east-southwest direction.

Bordering the quartz-monzonite are contact deposits, replacement

¹ J. M. Boutwell: *Economic Geology of the Bingham Mining District*, Utah. *Professional Paper No. 38*, U. S. Geological Survey (1905).

J. M. Boutwell: *Genesis of the Ore-Deposits at Bingham, Utah*. *Trans.*, xxxvi, 541 to 580 (1905).

deposits in the limestone lentils, and fissure veins; but the orebody producing more copper than the combined output of all other mines in the district is located in one of the larger masses of quartz-monzonite and is of the disseminated type.

e. The Utah Copper Mine.—In the mass of monzonite porphyry, locally termed the Bingham laccolite, forming the larger part of a mountain, about 1,600 ft. high, between Bingham and Carr Fork Canyon, the development work of the Utah Copper Co. has proved the existence of an enormous orebody. A description of this property and its development is given in the following quotation, from a report issued in August, 1914:²

"The orebody of the Utah Copper Co. consists of an altered siliceous porphyry, containing small grains of copper minerals, very uniformly disseminated throughout the mass, both in fracture seams and in the body of the rock, averaging about 1.5 per cent. copper, 0.15 oz. of silver and 0.015 oz. of gold.

"The total area of lode mining claims in Bingham owned by the Utah Copper Co. is 736 acres, within the boundaries of which development has shown that at least 225 acres contain mineralized porphyry of commercial value. The entire porphyry area has not yet been developed; and the maximum thickness of the orebody in the 225 acres has not been fully demonstrated, but the existing development in this area shows now an average thickness of 445 ft., which is equivalent to about 1,500,000 tons of ore per acre, or a total of about 361,220,000 tons. In making the calculations to determine the tonnage of ore and its average assay, there were used 50,761 assays, representing 23,465 ft. of diamond-drill and churn-drill holes, 285,913 ft. of drifts, raises, and winzes, and 7,130 linear feet of steam-shovel cuts, or a total of 316,508 linear feet of development work. The orebody, as at present developed, has a maximum length of a little over one mile and a maximum width of more than one-half mile. Further development of the property will add materially to its ore reserves."

From 1906 to 1913 this property was developed on a large scale by steam-shovel and underground methods. In 1913, the underground work was discontinued, and since that time the entire production has been handled with the steam shovels.

The steam-shovel output has increased from 300 tons of ore daily in 1906 to 22,000 tons daily in 1914. At present 21 steam shovels are engaged in the development work. Some of the shovels are working in ore and others are removing capping. In the future the number of shovels working in the ore will be increased, while those working in capping will be decreased; thus the daily production will gradually increase as the work proceeds.

f. Field and Laboratory Work.—In 1912, the writer was employed by the Utah Copper Co. for six months as underground sampler, and in the summer of 1914 three months were spent on the surface. During this time a suite of about 300 specimens was collected. This collection in-

² American Institute of Mining Engineers: One hundred and eighth meeting, Salt Lake City, Utah, Aug. 10 to 14, 1914.

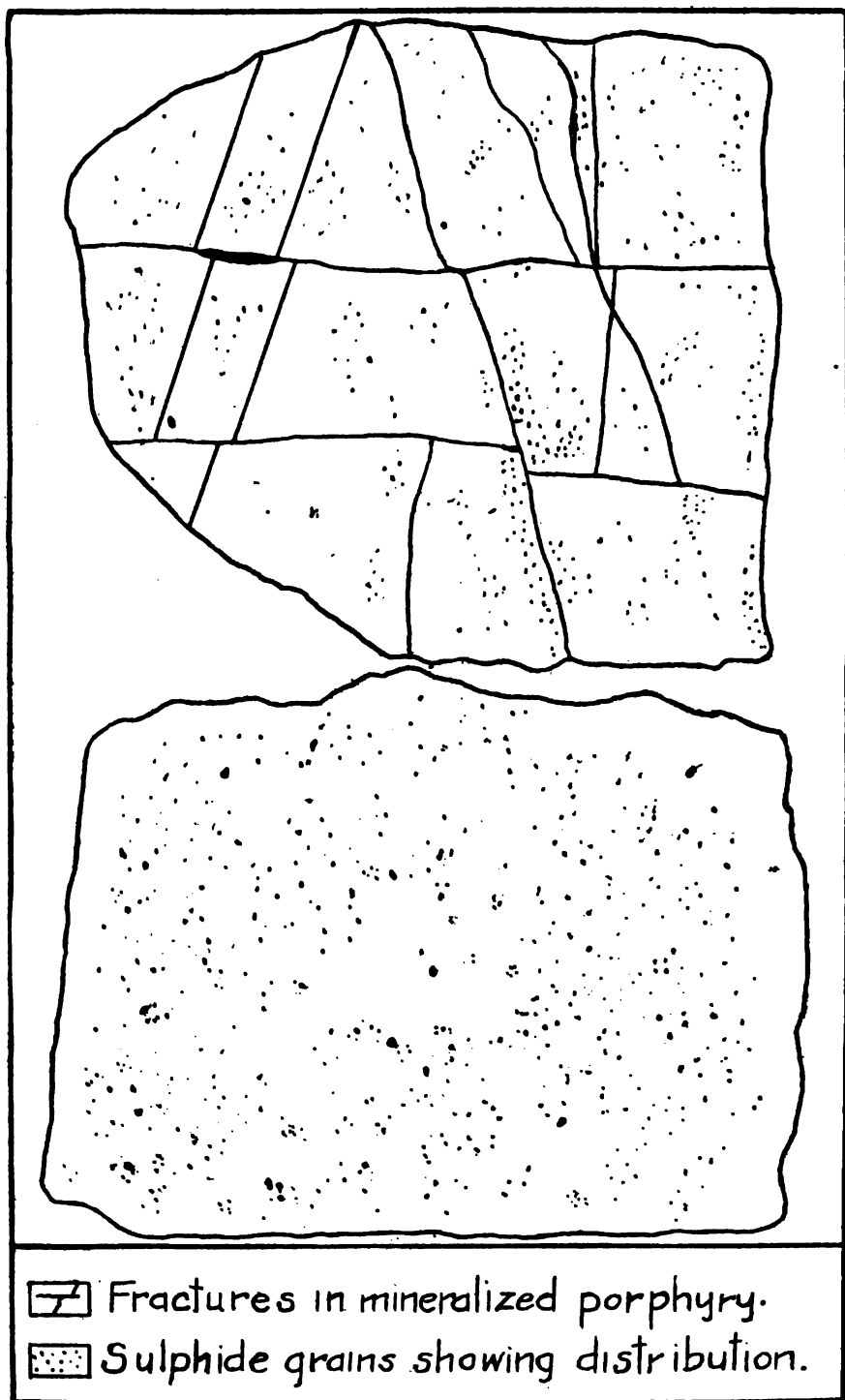
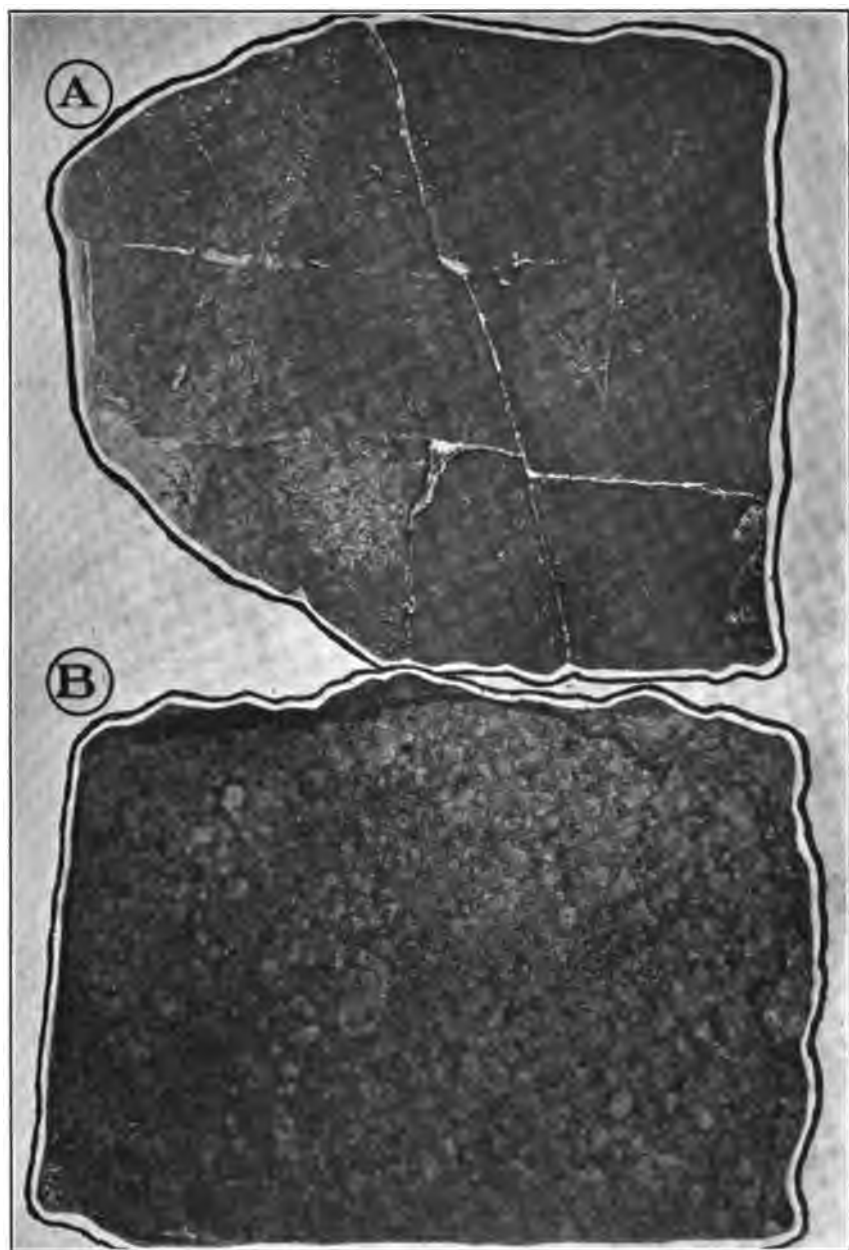


PLATE I



Mineralized Quartz-Monzonite from Utah Copper Underground Mine. Natural size.
PLATE I

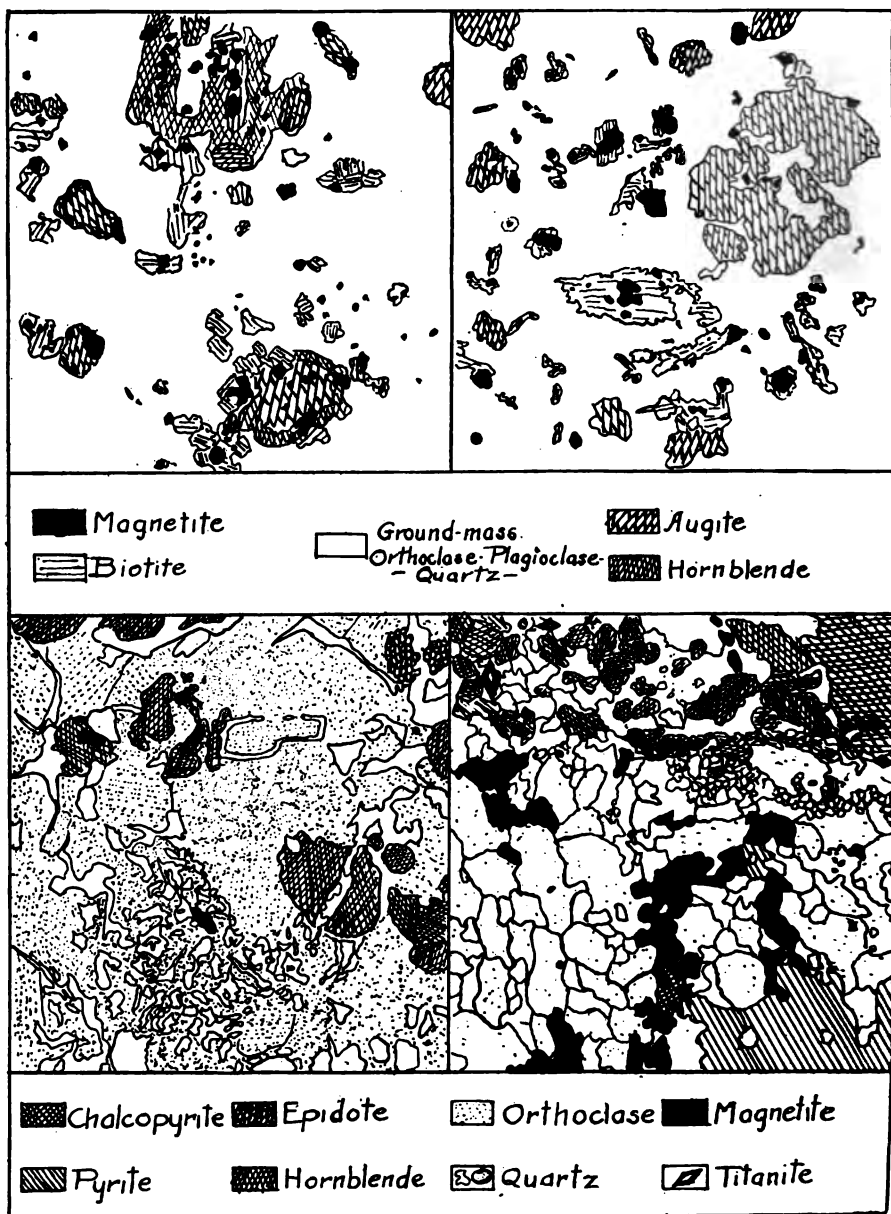
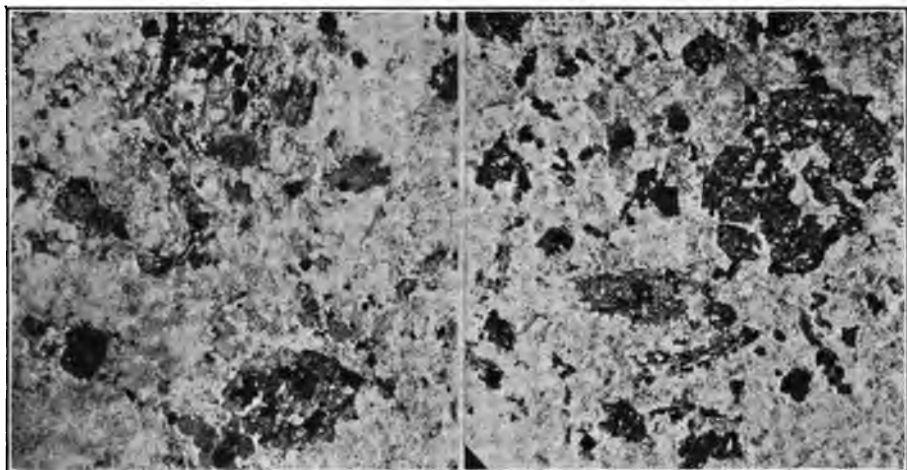
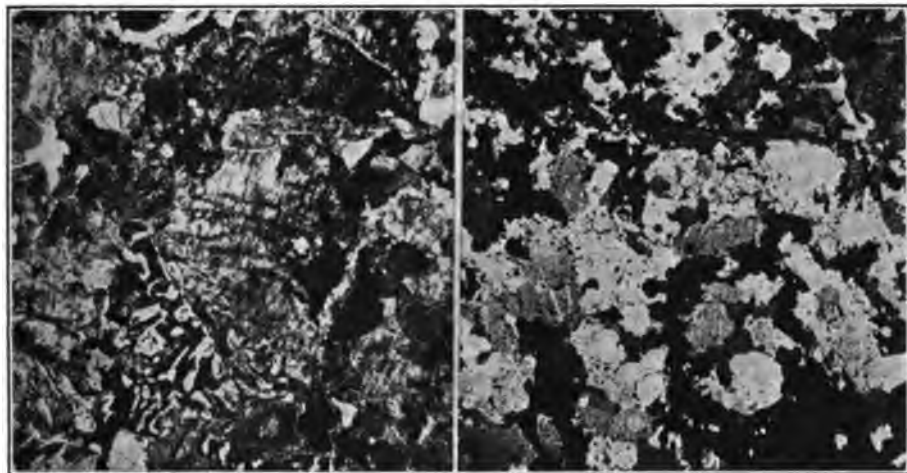


PLATE II



A.—Quartz-Monzonite from near Silver Shield Mine. $\times 10$ diameters. Ordinary light.

B.—Quartz-Monzonite from near Highland Boy Mine. $\times 26$ diameters. Ordinary light.



C.—Pegmatite Vein from Commercial Mine. $\times 6$ diameters. Parallel nicols.

D.—High-Temperature Quartz Vein from Commercial Mine. $\times 27$ diameters. Crossed nicols.

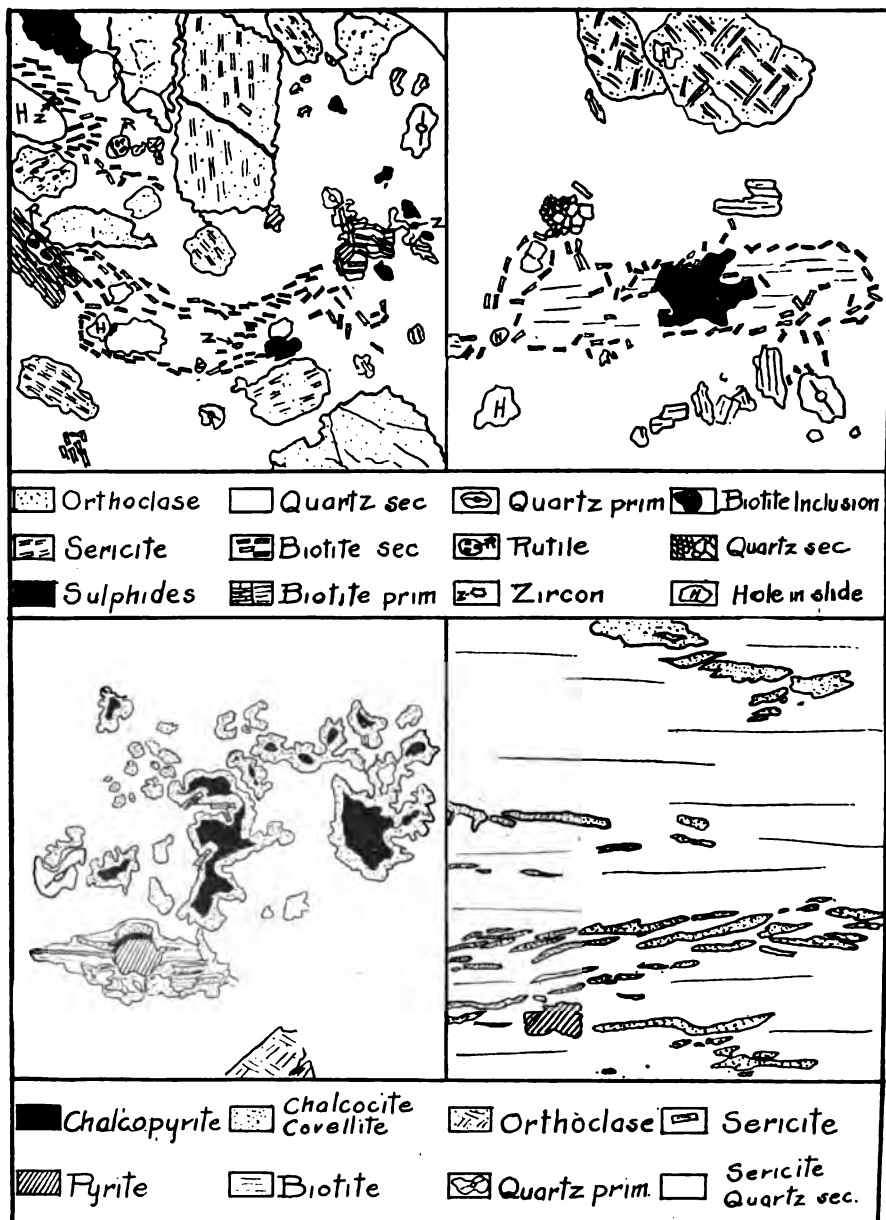
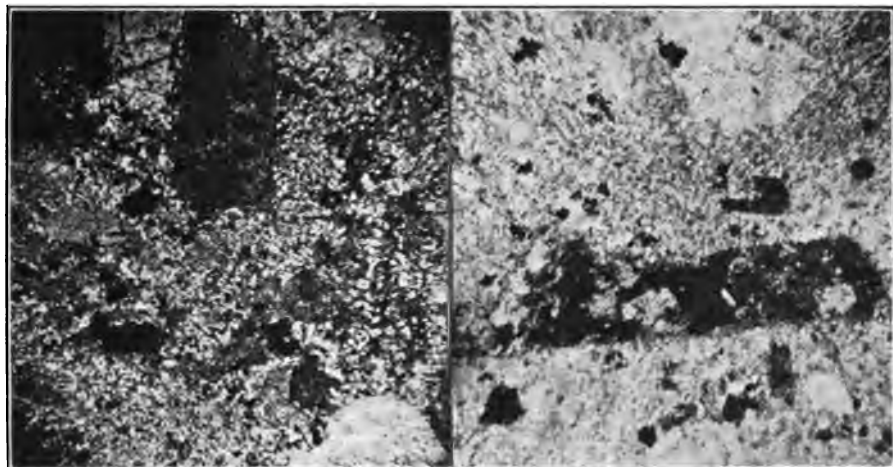
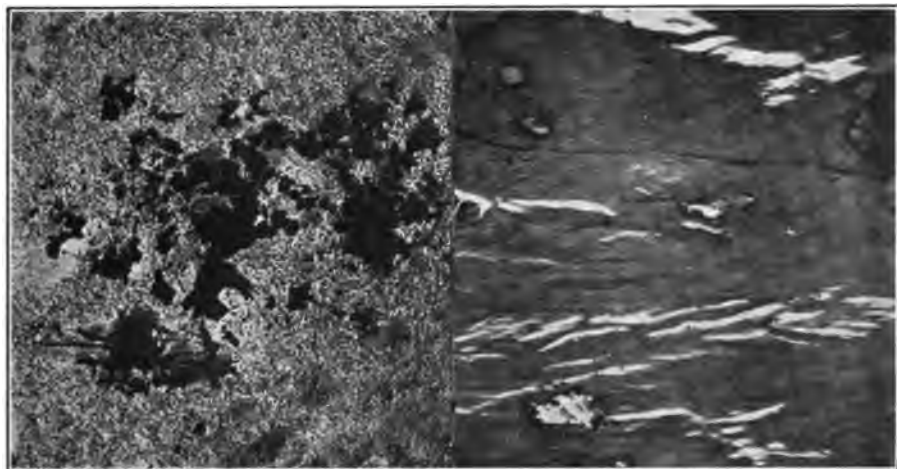


PLATE III



A.—Metasomatic Replacement of Feldspar Groundmass by Secondary Quartz. Mineralized Quartz-Monzonite. $\times 8$ diameters thin section. Crossed nicols.

B.—Chalcopyrite in Altered Crystal of Biotite. From hand specimen B, Plate I. $\times 12$ diameters thin section. Crossed nicols.



C.—Pyrite Grains in Biotite, Chalcocite, Covellite, along Cleavage Planes. Mineralized Quartz-Monzonite. $\times 51$ diameters thin section. Ordinary light.

D.—Enlargement from Same Specimen as C, showing Chalcocite as Gashes in Biotite Crystal. $\times 100$ diameters. Polished section.

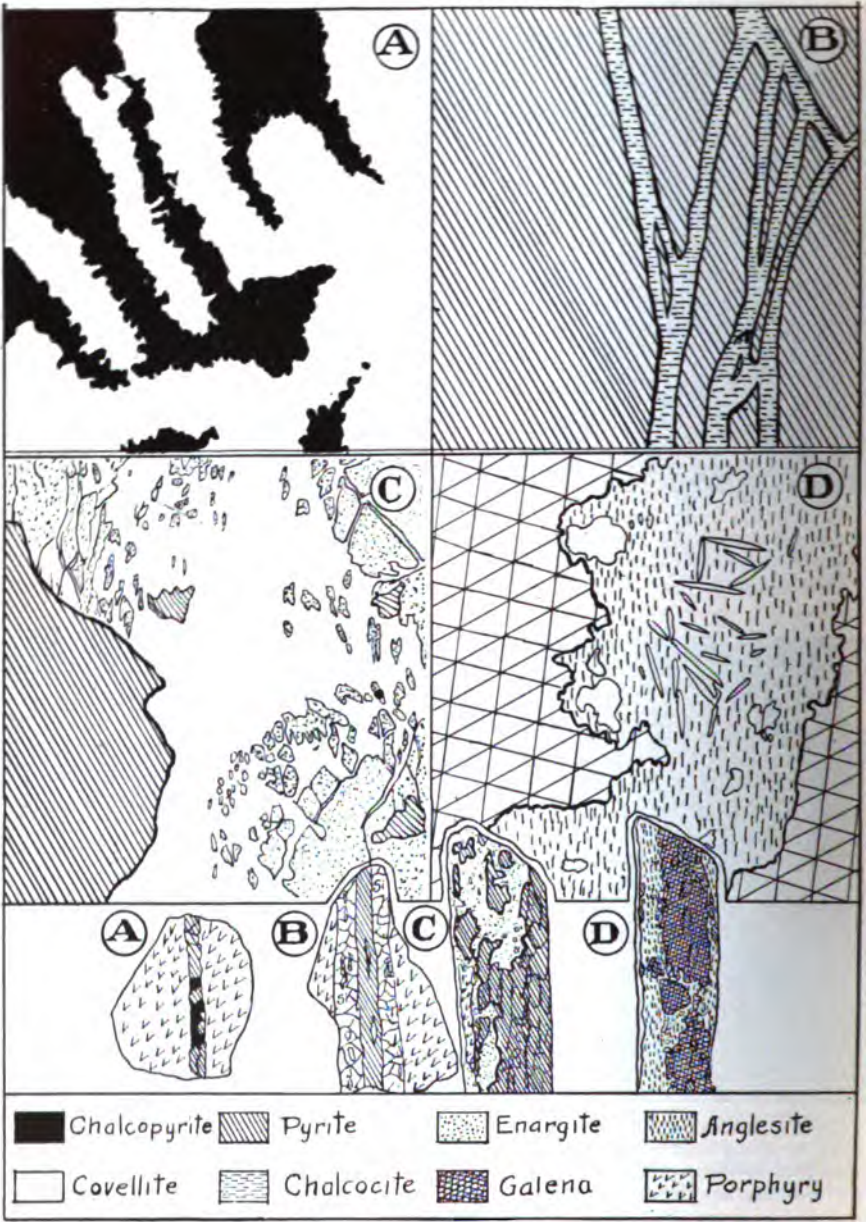
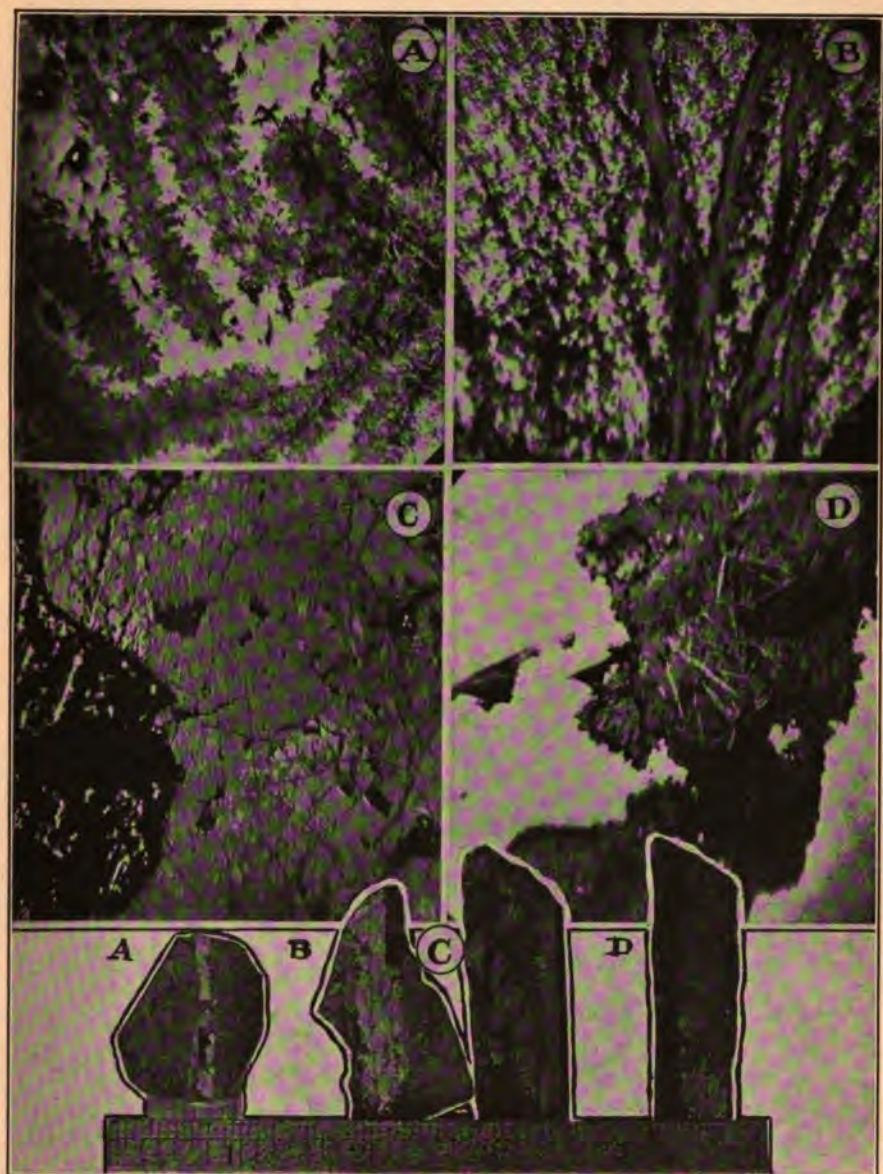


PLATE IV



A.—Covellite Replacing Chalcopyrite. $\times 96$ diameters.
 B.—Chalcocite Replacing Pyrite. $\times 241$ diameters.
 C.—Covellite Replacing Enargite. $\times 72$ diameters.
 D.—Covellite Replacing Galena. $\times 74$ diameters.

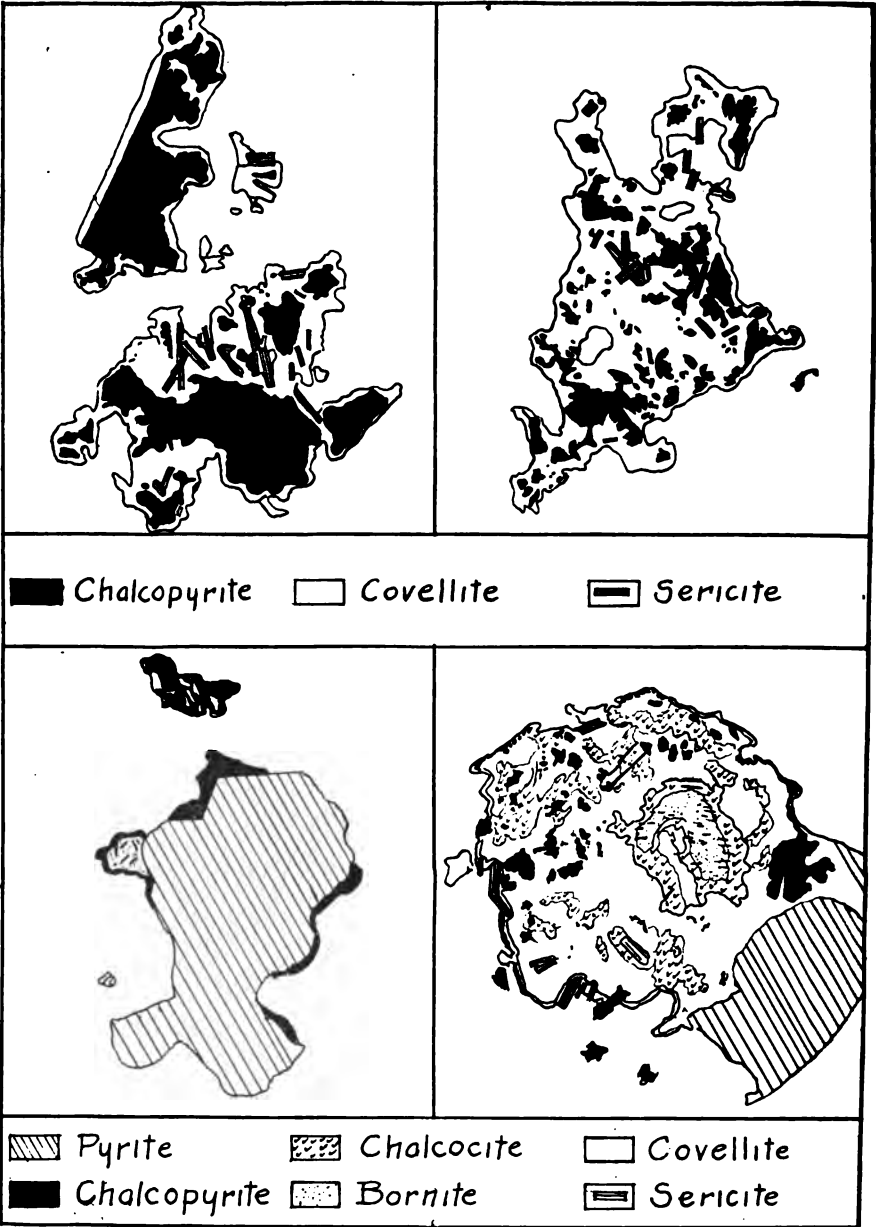


PLATE V



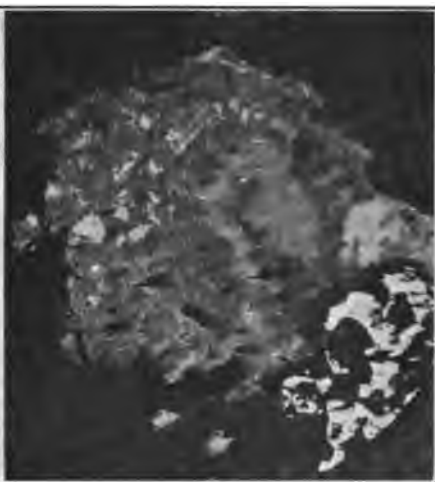
A.—Covellite Replacing Chalcopyrite. Covellite Ores, Underground Mine. $\times 129$ diameters. Polished section.



B.—Covellite Replacing Chalcopyrite. Covellite Ores, Underground Mine. $\times 103$ diameters. Polished section.



C.—Pyrite, Chalcopyrite, Bornite. Covellite Ores, Underground Mine. $\times 96$ diameters. Polished section.



D.—Pyrite, Chalcopyrite, Bornite, Chalcocite, Covellite. Covellite Ores. $\times 55$ diameters. Polished section.

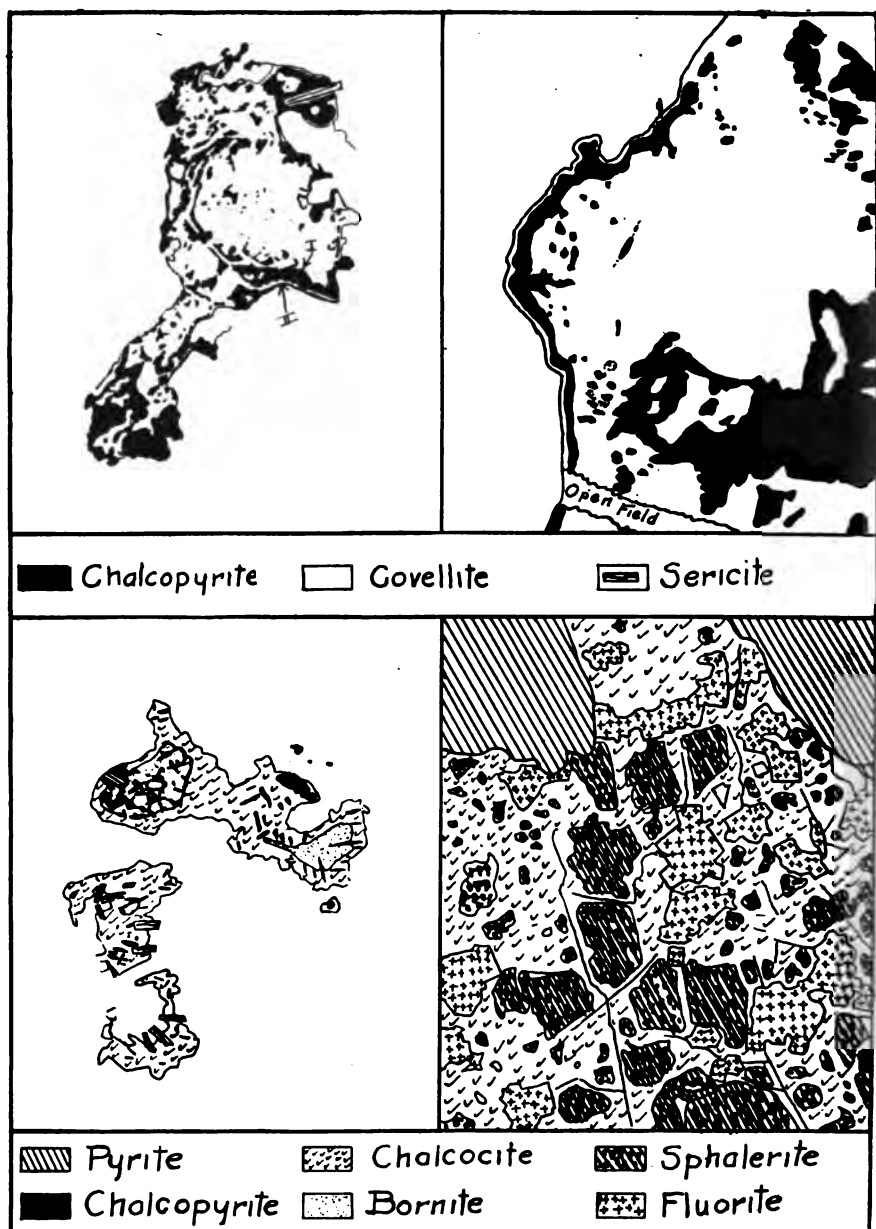


PLATE VI



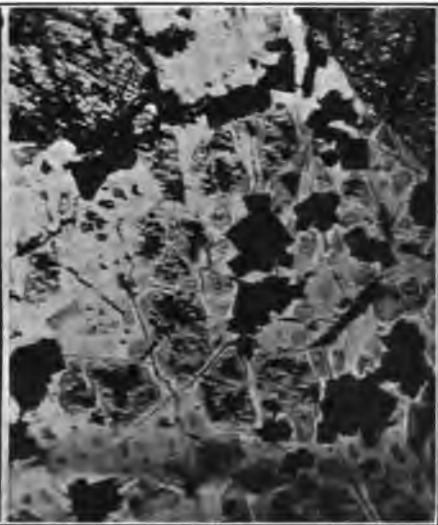
A.—Chalcopyrite Replacing Covellite. Covellite Ores. $\times 96$ diameters. Polished section.



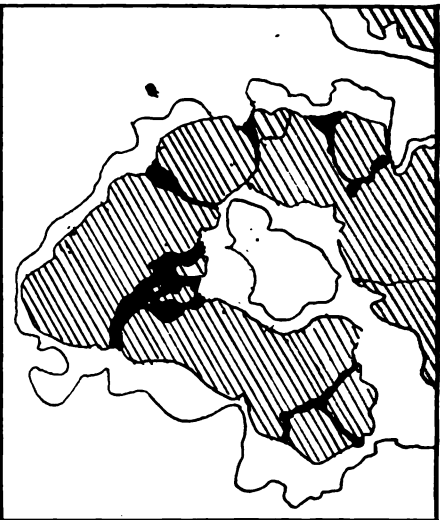
B.—Chalcopyrite Replacing Covellite. Covellite Ores. $\times 212$ diameters. Polished section



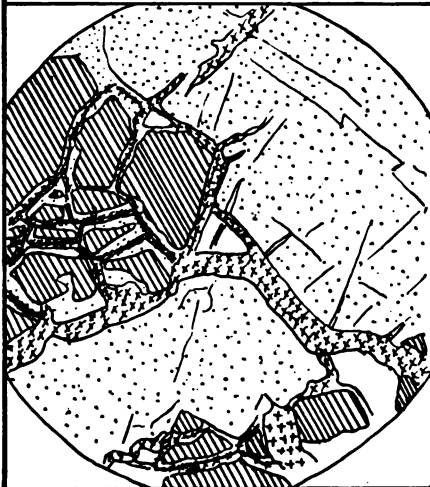
C.—Chalcocite Replacing Bornite. Covellite Ores. $\times 55$ diameters. Polished section.



D.—Chalcocite Replacing Sphalerite. Covellite Ores. $\times 96$ diameters. Polished section.



- | | | |
|--|--|--|
|  Pyrite |  Chalcocite |  Melaconite |
|  Chalcopyrite |  Bornite |  Malachite |



- | | | |
|--|--|--|
|  Pyrite |  Chalcocite |  Chalcopyrite |
|  Melaconite | |  Covellite |

PLATE VII



A.—Chalcocite Replacing Chalcopyrite with Bornite as an Intermediate Product. Chalcocite Replaced by Melaconite. $\times 58$ diameters. Polished section.



B.—Chalcocite Replacing Pyrite and Chalcopyrite. Chalcocite Ores. $\times 116$ diameters. Polished section.



C.—Chalcocite Replacing Pyrite. Melaconite Replacing Chalcocite. Chalcocite Ores. $\times 55$ diameters. Polished section.



D.—Covellite Replacing Chalcopyrite. Chalcocite Replacing Covellite and Chalcopyrite. $\times 64$ diameters. Polished section.

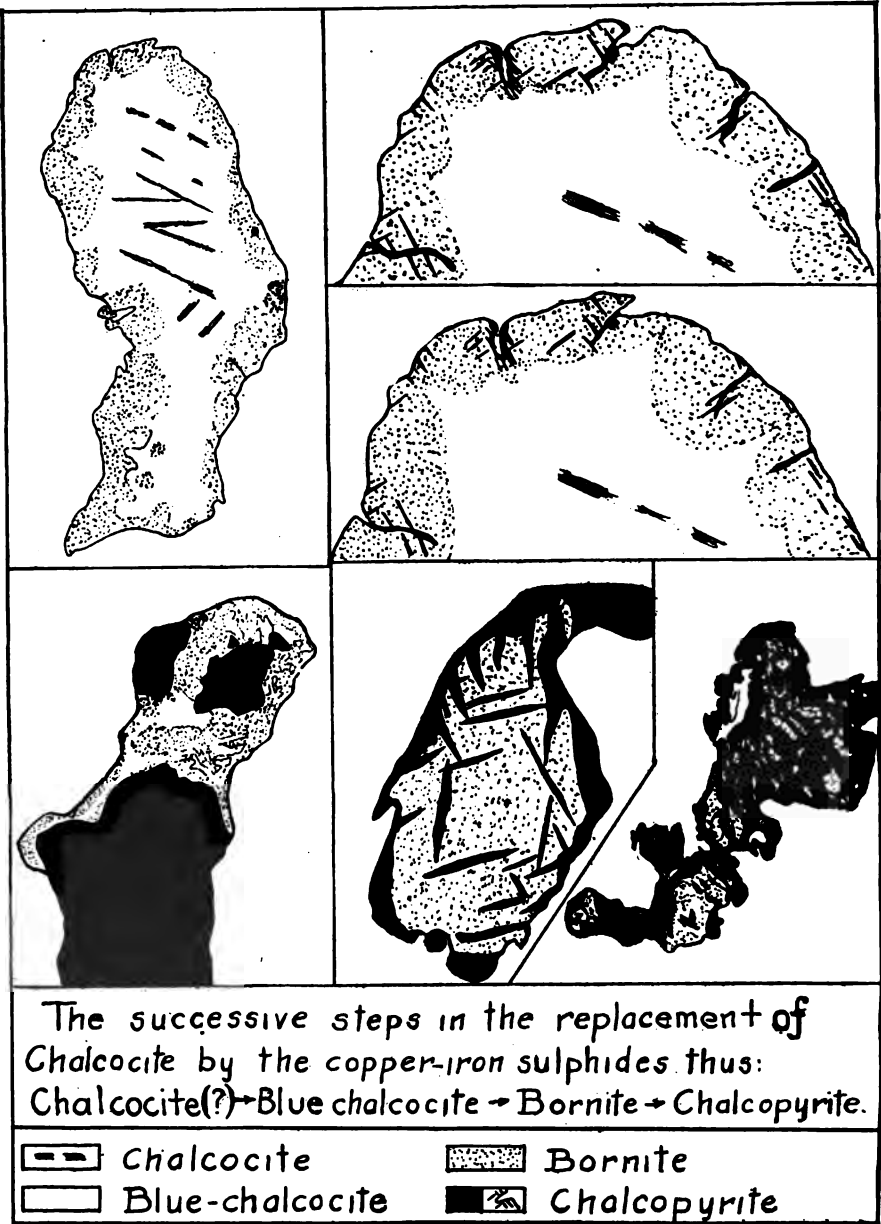
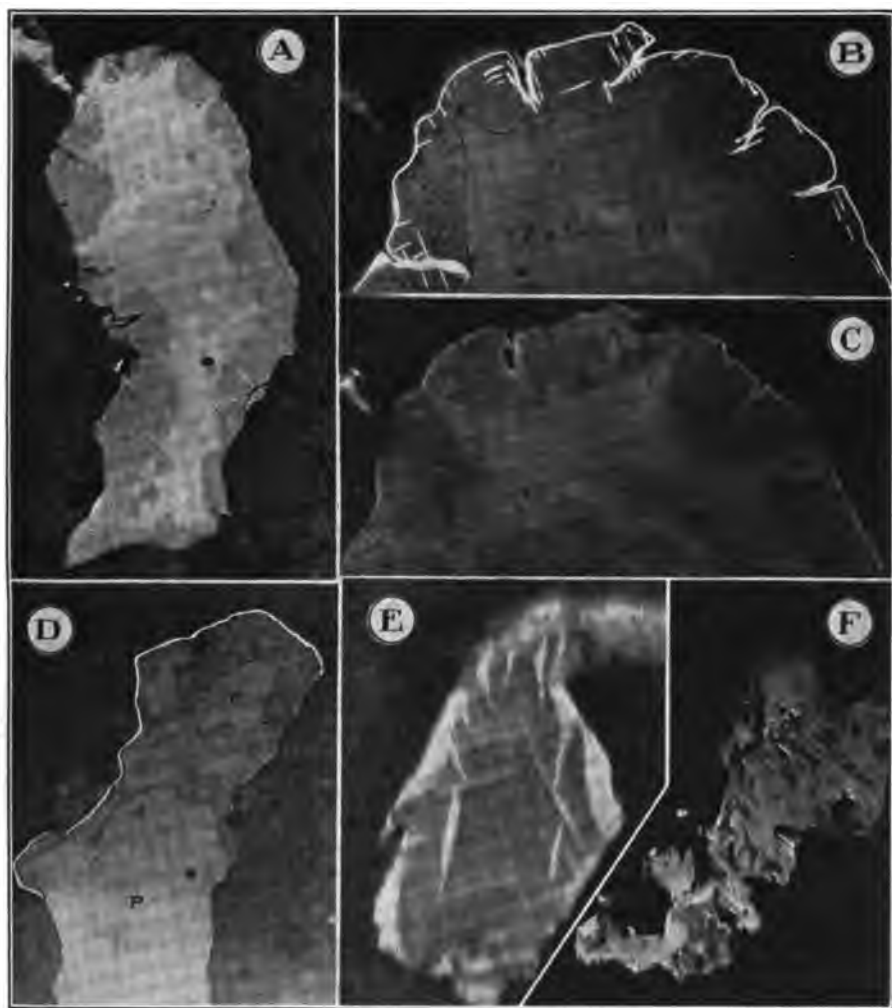


PLATE VIII



Chalcopyrite Replacing Chalcocite, as Borders and Parallel Gashes, showing Partial to Almost Complete Replacement.

A.—Partial Replacement of Chalcocite by Bornite and Chalcopyrite. $\times 111$ diameters.

B, C.—Enlargements of A. Chalcocite Ores. $\times 341$ diameters.

D.—Primary and Secondary Chalcopyrite. $\times 63$ diameters.

E.—Secondary Chalcopyrite Replacing Secondary Bornite. $\times 431$ diameters.

F.—Secondary Chalcopyrite almost completely Replacing Bornite and Chalcocite. $\times 95$ diameters.

PLATE VIII

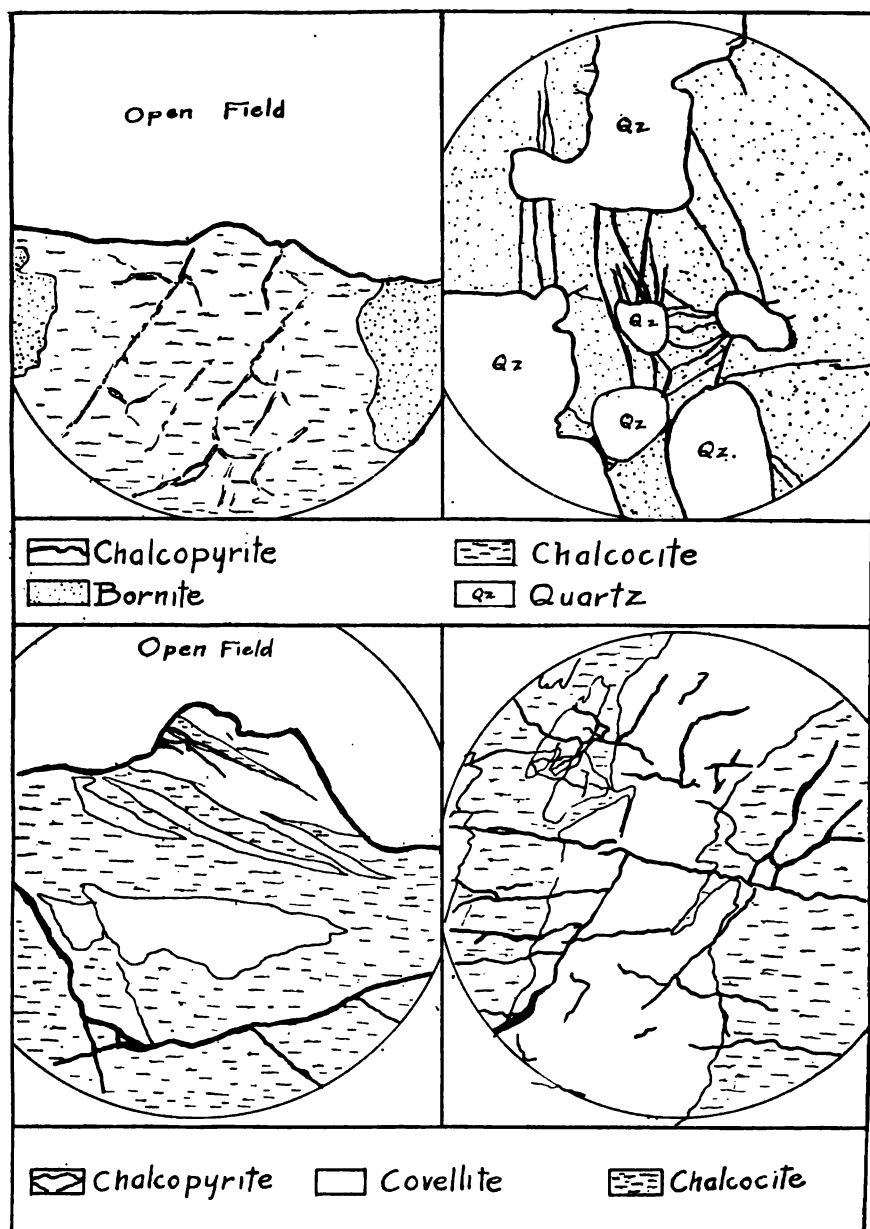


PLATE IX



A.—Synthetic Chalcopyrite Replacing Chalcocite. $\times 57$ diameters. Polished section.



B.—Synthetic Chalcopyrite Replacing Bornite in Veinlets and Bordering Quartz Grains. $\times 51$ diameters. Polished section.



C.—Synthetic Chalcopyrite Replacing Chalcocite and Covellite. $\times 59$ diameters. Polished section.



D.—Synthetic Chalcopyrite Replacing Covellite and Chalcocite. $\times 55$ diameters. Polished section.

PLATE IX

Published by courtesy of Prof. S. W. Young.

cludes surface and underground samples of the quartz-monzonite from many parts of the district in and outside the area of mineralized porphyry.

From a selection of the material collected the writer has made and studied—in the Laboratory of Economic Geology at Stanford University—70 thin sections and more than 100 polished sections, with the view of determining the possible stages through which the copper has passed from the time it existed in the earliest magma to its present condition as disseminated copper minerals, enriched by secondary processes.

2. *The Quartz-Monzonite*

The quartz-monzonite in the hand specimen is usually a dark-gray, fine-grained, or light-gray porphyritic rock, containing locally a few exceptionally large phenocrysts of orthoclase— $\frac{1}{2}$ to 1 in. long. *A* and *B*, Plate I, are photographs (natural size) of hand specimens of the mineralized porphyry. The photographs show both the fine and coarse textures of the porphyry, even after intense silicification. The finely crystalline porphyry—*A*—predominates throughout the district; but locally the coarse-textured rock may be present. In general, thin sections of the unaltered quartz-monzonite show uniformity in mineral constituents regardless of the texture.

The chief minerals of the quartz-monzonite, named in the order of their abundance, are orthoclase, and plagioclase, occurring in well-developed phenocrysts and in the granular groundmass; biotite, in phenocrysts and in small flakes; augite, often surrounded by alteration rims of uraltic hornblende (*A*, Plate II), and interstitial quartz.

The accessory minerals are magnetite, in large and small irregular grains, and rutile, occurring as sagenite webs in biotite. Apatite and zircon are present in very small well-formed crystals.

A and *B*, Plate II, are photographs of thin sections of the quartz-monzonite. *A* is from a sample collected on the mountains above the Silver Shield mine; *B* is from an outcrop near the Highland Boy mine. These localities are about 1 mile apart and both are about 1 mile from the disseminated ore deposit. In *A* the plagioclase is more prominent than orthoclase; while in *B* orthoclase is the chief mineral in the groundmass. The illustrations show the principal minerals; also the association of biotite with magnetite.

The interesting features of the rock are the association of biotite with the magnetite, and the occurrence of interstitial quartz.

The magnetite is present in larger amounts than is common in igneous rock of the monzonitic type. It occurs as large and small anhedral grains, often in biotite crystals or surrounded by a narrow margin of small crystals of this mineral. A careful study of these margins shows that they have assumed their present position later than the solidification of the feldspar groundmass. There are two possible interpretations of

this phenomenon: (1) the biotite was formed by a reaction between the magnetite and certain late magmatic products; (2) the magnetite is of a late magmatic period and is replacing the biotite. The fact that the same phenomenon was observed in slides from the Sudbury district and was evidently caused by a partial replacement of the biotite by the sulphides, inclines the present writer to favor the second interpretation. In either case, this occurrence of biotite and magnetite explains the apparent replacement of biotite by the sulphides in the hydrothermal period, as will be explained later.

The interstitial quartz occurs in spaces about previously formed crystals and was the last silicate mineral to solidify. It is present in all slides, but usually not in large amounts. However, when the amount is large, certain changes take place; in some slides the phenocrysts of feldspars appear to have been attacked and changed into the finer-grained granular feldspars of the groundmass, while the augite shows alteration rims of hornblende. A further concentration of this quartz in the form of small pegmatite veins produces similar changes but to a greater degree.

3. *The Pegmatite Veins*

Specimens of porphyry from the Commercial mine—outside the area of mineralized porphyry—contain small white veinlets, $\frac{1}{2}$ in. in thickness, composed of quartz and feldspar which are visible to the naked eye. The large amount of quartz in the country rock gives it the appearance of a granite rather than of the typical quartz-monzonite.

Examination of the thin section under the microscope shows an increase in the interstitial quartz, accompanied by the formation of a graphic intergrowth of quartz and orthoclase, as shown in *C*, Plate II. This intergrowth occurs on the edge of the veinlet, while the main body of the vein is composed of irregular grains of quartz and orthoclase. The pyroxene has been completely changed to hornblende, which still possesses some of the petrographic properties of the original pyroxene. Biotite has almost completely disappeared and epidote is present instead. Rutile is absent and titanite seems to have taken its place. The magnetite and pyrite are present in separate and connected anhedral grains but not in amounts that would indicate a considerable addition of iron. The magnetite appears to have been attacked and it is possible that hydrogen sulphide was responsible for the change of magnetite to pyrite. The polished section shows the presence of a very small amount of chalcopyrite.

The relation between the pegmatite and the high-temperature quartz vein was not apparent in the field, so only a limited number of specimens were taken; however, the similarity in the mineral constituents suggests a close relation between the two.

4. *The High-Temperature Veins*

A sample from the same location as the pegmatite veins—according to Lindgren's Mineral Classification—proved upon microscopic examination to be a high-temperature vein.

The vein was composed of massive quartz containing a considerable amount of metallic sulphides. A narrow strip of green hornblende separated the quartz and the country rock.

Examination of the thin section showed the hornblende strip to be composed of hornblende, formed from pyroxene, together with considerable amounts of epidote, titanite, and apatite. The main body of the vein is composed of quartz in which magnetite and pyrite, with a small amount of chalcopyrite, occur in such relations that they do not appear to have replaced any of the silicate minerals or to have replaced one another. It is very probable that they assumed their present form upon the solidification of the quartz.

An interesting feature of the quartz is that many of the crystals are clear in the center but in the outer zone contain many small inclusions which appear to be apatite and zircon. *D*, Plate II, shows the contact between the hornblende strip and the quartz; also the occurrence of the minerals which have been described.

In the Silver Shield mine and also in the Sulphide mine, near the Highland Boy, samples of the porphyry near certain veins showed alterations similar to those which have just been described, except quartz, titanite, and apatite, which were practically absent, while biotite and epidote were both present. In some slides the biotite and pyroxene were both altering to chlorite.

The pyrite and chalcopyrite are not magmatic sulphides, because secondary quartz, epidote, or other minerals,—which indicate the action of mineralizers,—are present in the same slide.

The vein minerals and rock alterations which have been described are common in the high-temperature veins, in the quartz-monzonite, in different parts of the district, but are not found in the disseminated ores; furthermore, the alterations which are found in the disseminated ores are characteristic of hydrothermal conditions; thus there is little doubt that the disseminated ores were deposited by heated aqueous solutions rather than under gaseous or magmatic conditions.

The pegmatite and the high-temperature quartz vein, from the Commercial mine, may be considered as rock extracts, and indicate, to a certain degree, the composition of the mineralizers which were active during the solidification of the magma.

The processes which have been effective in the formation of these

mineralizers and also in the bringing about of the alterations which have been described are clearly given in the following words by Harker.²

"Pneumatolytic Action.—With the completion of crystallization in a rock-magma the contained water and other volatile substances, excepting such portion as has been incorporated in some of the crystallized minerals, must be disengaged. Since the critical temperature for water is about 365 degrees centigrade, and for other substances lower, they must be in the gaseous state, however great the pressure to which they may be subjected.

"The active rôle of the volatile constituents does not terminate with the completion of the crystallization. Having fulfilled the constructive office of mineralizers, they now enter upon a new activity which is partly of a destructive kind. As cooling proceeds, some of the compounds crystallized at higher temperatures, cease to be stable in the presence of the concentrated gaseous residuum, and are decomposed with the production of new minerals. The volatile substances themselves may or may not enter into the composition of these new minerals, but in general they do so to a greater extent at this stage than during the magmatic crystallization.

"Pneumatolytic action in plutonic rocks is not confined to their pegmatoid modifications. In many cases it clearly follows joint-fissures in the body of the rock, sufficient proof that it belongs to a stage distinctly posterior to the completion of consolidation."

At lower temperatures these gases become liquids and the changes due to hydrothermal activity follow.

5. *The Early Hydrothermal Period*

The early part of the hydrothermal period marks the most important period of mineralization in the disseminated ores, because the solutions in this period caused the comparatively uniform dissemination of chalcopyrite and pyrite in an area more than 1 mile long and in some places more than $\frac{1}{2}$ mile wide.

Although faulting and intense fracturing in the rock are widespread throughout the district it does not appear that in the disseminated area the early mineral-bearing solutions were confined to fractures or fissure veins, but, on the contrary, these solutions have caused exceptionally uniform distribution of fine-grained secondary quartz and sulphide grains throughout the entire mass of the rock in large areas.

The following quotation from Mr. Boutwell's report describes clearly the mineralized quartz-monzonite of the disseminated ores and also suggests the possible relation between the magnetite, chalcopyrite, and pyrite.

"A specimen of highly altered monzonite has lost the dark color and compact body and shows instead a dull, light gray color, a slightly porous structure, and abundant quartz in veinlets and blotches upon the walls of parting planes. Conspicuous areas of granular quartz are numerous, the orthoclase is highly sericitized and feldspar minerals

² Alfred Harker: *The Natural History of Igneous Rocks*, p. 266 (1909).

are represented by numerous irregular patches of small individuals or flakes of dense brown biotite. The quartz and sericite are clearly secondary and though no direct proof of the age of the biotite has been found, it resembles secondary biotite and may be secondary also. Magnetite, excepting occasional grains, has disappeared, and large amounts of chalcopyrite and pyrite are present in the form of rounded grains, chains, and veinlets embedded in secondary quartz, flaky biotite, and sericitized feldspar.

"The observed stage of metasomatic alteration of magnetite, culminating in the occurrence of minute, ill-defined cores of magnetite without secondary sulphides and finally in the total disappearance of magnetite, indicates one source of iron of the chalcopyrite and pyrite. Additional iron was doubtless derived from original augite and biotite. Any additional sulphur which may have been required was probably supplied from without, perhaps in the form of sulphurous gases."

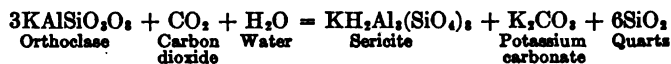
The alterations just described and the conclusions drawn in this quotation are in harmony with the present paper.

A detailed study indicates that the first alterations were as follows: The feldspar groundmass in large areas of the quartz-monzonite has suffered complete metasomatic replacement by a very fine-grained secondary quartz, the feldspar phenocrysts have been partly, and in some cases completely, replaced by this secondary quartz; but in spite of this intense silicification some of the remaining feldspar fragments are perfectly fresh and free from sericite, while others have varying amounts, which may have been formed later than this period. The sericite which is present in the slides showing the least amount of this mineral is confined to the feldspars and does not replace the secondary quartz. There are two possible explanations for the small amount of sericite: (1) secondary biotite may have been formed instead of sericite; (2) the amount of carbon dioxide in the first period of mineralization was either small or inactive.

1. In the mineralized monzonite least affected by sericitization there are usually large amounts of small green and brown crystalline aggregates of secondary biotite. This biotite may have formed instead of sericite because of the excess of iron present, part of which was probably in the form of more or less complex molecules which formed in the break-down from the original augite, which in this altered rock has completely disappeared, leaving no direct evidence as to its alteration products. In this case the secondary biotite may have the formula of sericite in which part of the aluminum has been replaced by iron, giving the formula $\text{Al}_2\text{Fe}(\text{SiO}_4)_3\text{KH}_2$, described by Clarke.⁴ However, in many slides the secondary biotite is clearly derived from the primary biotite—*B*, Plate III, which is broken up into smaller crystals, many of which are still oriented in parallel position in the space occupied by the primary crystal, and probably have the same relation to the primary biotite as sericite has to muscovite.

⁴ F. W. Clarke: The Constitution of the Natural Silicates, *Bulletin No. 588*, U. S. Geological Survey, p. 51 (1914).

2. The formation of sericite often assumes the presence of carbon dioxide; its formation from orthoclase may be represented by the following equation:⁶



In the early hydrothermal period the amount of carbon dioxide was either small or in the presence of the other components of the solution it was not effective in producing sericitization. Lindgren in speaking of volcanic gases says:⁶

"Among the volcanic gases nitrogen, carbon dioxide, and hydrogen sulphide are the most important and their emission, particularly that of carbon dioxide, may continue long after the igneous activity has subsided. . . . Evidence of this is furnished by the exhalations of carbon dioxide and nitrogen in the mines at Cripple Creek, of nitrogen at Creede, and of carbon dioxide in the Tertiary gold deposits of New Zealand."

The alterations in the disseminated ores give strong evidence that sericitization for the most part has been effective after the formation of the fine-grained secondary quartz and also after the deposition of the disseminated primary sulphides, for in many of the slides the feldspars are completely replaced, and small lath-shaped crystals of sericite penetrate into the chalcopyrite (*A* and *B*, Plate V), but seldom into the pyrite grains; thus the sericite which is present in the slides showing the least amount of this mineral is probably of a later period than the first alteration. The author favors the second explanation.

The replacement of the sulphides by sericite was first described by Rogers in a magmatic ore deposit in Plumas County, Cal.⁷ The occurrence in Bingham presents a parallel case.

A, Plate III, is a microphotograph showing the irregular cross-sections of comparatively fresh feldspar phenocrysts imbedded in a groundmass of fine-grained secondary quartz. The accompanying outline plate—drawn from the slide—shows the distribution of the secondary biotite and also the other minerals which are present in the rock containing the least sericite.

The sagenite webs of rutile may still be seen in the fragments of primary biotite, but as the alteration proceeds these crystals separate out in small crystalline aggregates and in larger crystals which are evidently formed by recrystallization.

⁶ F. W. Clarke: The Data of Geochemistry, *Bulletin No. 491, U. S. Geological Survey*, p. 512 (1911).

⁶ Waldemar Lindgren: *Mineral Deposits*, p. 445 (1913).

⁷ H. W. Turner and A. F. Rogers: A Geologic and Microscopic Study of a Magmatic Copper Sulphide Deposit in Plumas County, California, and Its Modification by Ascending Secondary Enrichment. *Economic Geology*, vol. ix, No. 4, p. 386 (June, 1914).

The pyroxene and hornblende have completely disappeared. Occasionally in the porphyry outside of the mineralized areas magnetite appears to form when the pyroxene is undergoing alteration; it is therefore probable that some of the iron present in the sulphides was derived from this source.

Zircon and apatite are more noticeable than in the unaltered monzonite, but are probably unaltered minerals which were present in the original rock.

Magnetite has almost completely disappeared and in its place there are chalcopyrite and pyrite, the distribution of which bears a striking similarity to that of the magnetite in the unaltered quartz-monzonite.

Occasionally pyrite or chalcopyrite occurs in altered crystals of the primary biotite in such a way that the replacement of biotite by the sulphides is suggested. *B*, Plate III, is a good example of this apparent replacement, but it will be noted here that a small crystal of secondary biotite, showing no signs of replacement, is present as an inclusion in the sulphide. These inclusions are not uncommon in grains of chalcopyrite and occasionally in pyrite. In some slides the secondary biotite cuts the sulphide grains and also the secondary quartz in such a way as to indicate that it is later than either. Considering the fact that secondary biotite is present in such large amounts forming simultaneously with and probably following the deposition of the sulphides, it is not reasonable to suppose that the biotite has been replaced by the sulphides.

The only conceivable way in which the sulphides could have been so evenly distributed, in a deposit of hydrothermal origin, would have been through the medium of a precipitating agent which was itself disseminated throughout the original quartz-monzonite. This precipitating agent was evidently confined to the quartz-monzonite because in the quartzite, which has been silicified by the action of the same solutions as have produced silicification in the quartz-monzonite, there is at most only a narrow margin which is mineralized.

In the Physical Chemistry Laboratory of Stanford University recent work has shown that alkaline solutions charged with hydrogen sulphide, alone or in the presence of carbon dioxide, under pressure are capable of carrying considerable amounts of metallic sulphides in colloidal suspension; furthermore, upon the escape of hydrogen sulphide the metallic sulphides are precipitated. A discussion of the experiments together with the geologic evidence supporting these results appears in a recent paper by Tolman and Clark.⁸

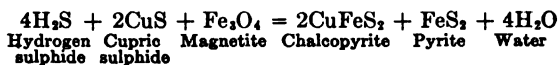
In *C*, Plate III, the sulphides appear along the cleavage planes of the primary biotite. The polished section *D*, Plate III, highly magnified,

⁸ C. F. Tolman, Jr., and J. D. Clark: The Oxidation, Solution, and Precipitation of Copper in Electrolytic Solutions, etc. *Economic Geology*, vol. ix, No. 6, pp. 559 to 592 (Sept., 1914).

shows the sulphides in the biotite to be pyrite in small grains, and covellite and chalcocite replacing the primary chalcopyrite which was deposited in small gashes along the cleavage planes of the primary biotite. Occurrences of this kind are rare but they certainly suggest a precipitating agent within the biotite.

Comparison of *C*, Plate III, with *B*, Plate II, shows the similarity in the occurrence of pyrite in biotite—in the mineralized area—to that of magnetite in biotite in the unaltered monzonite. Occurrences of this kind together with the similarity in the distribution of magnetite in the unaltered rock, and the sulphide in the mineralized monzonite, furnish good evidence that the magnetite is responsible for the precipitation of the sulphides, and also explain the apparent replacement of biotite by pyrite and chalcopyrite.

The following equation, though experimental work tends to prove it,⁹ is given merely to suggest the possible reactions which have taken place in the heated alkaline solutions carrying the copper and probably some iron as colloidal sulphides:



CuS is used here merely to balance the equation, but another copper or copper-iron sulphide could be substituted.

In the foregoing equation the removal of hydrogen sulphide from the solution and the formation of the less soluble sulphides chalcopyrite and pyrite, have both been factors in the precipitation. Magnetite has furnished iron for both chalcopyrite and pyrite, but there may have been an addition of iron at this time.

Another possibility is that the change of magnetite to pyrite by the action of hydrogen sulphide—proved experimentally by Doelter¹⁰ and others—has removed hydrogen sulphide from the solution, thus causing the precipitation of chalcopyrite directly. In either case there has evidently been a molecular rearrangement of the iron, because biotite inclusions are present in both pyrite and chalcopyrite.

In many of the slides and polished sections chalcopyrite and pyrite occur in separate grains in such a way as to indicate that the pyrite has not been replaced by chalcopyrite—for instance, in a certain area, say there are 12 sulphide grains; five may be composed wholly of pyrite, two of pyrite with a partial border of chalcopyrite, and five of solid chal-

⁹ Prof. S. W. Young. Personal communication: When chalcocite or covellite and magnetite are placed in the same test tube in slightly acid, alkaline, or neutral solutions in the presence of hydrogen sulphide under pressure the chalcocite or covellite is rapidly coated with chalcopyrite and the magnetite is probably changed to pyrite.

¹⁰ F. W. Clarke: The Data of Geochemistry, *Bulletin No. 491, U. S. Geological Survey*, p. 317 (1911).

copyrite, without any cores or residual fragments of pyrite in the chalcopyrite. These grains have no special arrangement with respect to veinlets or parting planes in the rock. Occurrences of this kind are considered evidence that in certain parts of the deposit chalcopyrite does not replace pyrite. Ores from the "Pit" do show a replacement of pyrite by chalcopyrite; this will be discussed later. After considering other evidence it is not at all impossible that chalcopyrite and pyrite were formed in many parts of the deposit at about the same time. Where chalcopyrite and probably a small amount of bornite appear with but little pyrite it is simply due to solutions richer in copper than those which have formed pyrite with less chalcopyrite and the direct change of magnetite to pyrite has taken place.

It is true that the rocks in all parts of the district are intensely shattered, but the uniformity in the distribution of the fine-grained secondary quartz throughout the mass of mineralized monzonite would suggest that if these fractures existed they were extremely small under the great pressure at depth. Outline Plate I, made from the photographs and polished sections, natural size, shows the distribution of the sulphide grains with reference to the small fractures.

The precipitation of sulphides in the way which has just been described favors a great vertical range for the deposition of the chalcopyrite and pyrite. The present development work of the Utah Copper Co.—surface and deep drilling—has proved this range to be a minimum of 2,500 ft.

6. *The Late Hydrothermal Period*

The late hydrothermal period is probably a continuation of the early period, occurring after a more intense fracturing and faulting throughout the district. Although this period was probably an important one in the mineralization of the lower-temperature replacement deposits and fissure veins bordering the monzonite, it has been of less importance in the disseminated deposit, because in the disseminated ores the small veinlets, though quite numerous, seldom exceed 2 or 3 in. in thickness and are never included in the regular mine samples. With these sulphides, covellite or chalcocite is generally present in such amounts that it is not possible to determine with certainty the order of formation of the different minerals.

Molybdenite is common as very thin coatings on slickensides and parting planes in the porphyry. This mineral is usually formed at high temperature and may have been formed in the early hydrothermal period.

Enargite, sphalerite, and galena are rare. To the writer's knowledge there are only four places in the entire area of mineralized porphyry where these minerals are found in veinlets.

Typical examples of some of the small veinlets together with enlargements from their polished sections are shown in A, B, C, and D, Plate IV.

The alterations accompanying and following this later period of mineralization have been silicification and sericitization along the parting planes in the rock. The intense silicification usually extends not more than an inch on either side of the veinlets in the black porphyry, but in the workings of the underground mine, located below the top of the mountain, a strip of monzonite about 200 ft. thick, bordering the quartzite, has been so intensely silicified and sericitized as to produce a light-colored rock which has been thought to be a separate intrusive mass. This light-colored rock is locally termed the "Payroll porphyry," after the Payroll group of claims, and hereafter will be called by that name. The Payroll porphyry outcrop forms the top of the mountain, where its resistance to weathering has governed the topography to a considerable degree.

The light color and the small brown specks visible to the naked eye in the hand specimen, serve to distinguish the Payroll from the "Black" porphyry—altered quartz-monzonite containing a large amount of primary and secondary biotite.

Examination of thin sections of Payroll porphyry proves beyond doubt that it is the same porphyry as the darker quartz-monzonite, which it grades into without a well-defined contact. The light color is due to the large amount of quartz and sericite, while the brown spots are shown to be small aggregates of rutile crystals occurring in bleached biotite or in the spaces once occupied by the primary biotite crystals.

While the later period may not have been important as one of mineralization, the sericitization in this period has been of great importance in increasing the porosity of the rock, in the Payroll porphyry throughout the entire mass, and in the Black porphyry along numerous small fractures in the rock, thus promoting uniform enrichment by surface waters.

II. SECONDARY ENRICHMENT

1. *Author's Note*

In the following pages the term secondary will be used in the sense of changes, alterations, and enrichment caused by descending surface waters or solutions.

The reader will keep in mind that the photographs of the polished sections in this paper, as in most of the recent papers, show areas which in the natural size are extremely small (compare sulphide grains, Plate I, with enlarged grains, Plate V, etc.) and thus the replacements shown are on a very small scale.

The conventional design on the outline plates indicating a certain mineral has been retained for that mineral, so far as possible, in all of the plates. The drawings have been made from the photographs and also by the use of the microscope and are therefore accurate. No attempt has been made

to show the different silicate minerals bordering the sulphide grains in the polished section, because determination of these minerals by reflected light is at best only a guess. In this paper determinations of silicate minerals have been made in thin sections. In the thin section the sulphide grains are usually surrounded by fine-grained secondary quartz (shown in *A*, Plate III), small flakes of green and brown biotite, and in most of the specimens a large amount of sericite is present.

2. *Factors Governing Enrichment*

The principal factors governing the enrichment of the disseminated copper ores in Bingham Canyon are as follows:

(a) The intense fracturing of the rock has formed numerous channels, affording uniform circulation of surface waters. *A*, Plate I, shows this fracturing, also faulting on a small scale. Often it is difficult to obtain hand specimens of the ordinary size without many small fractures running through them.

(b) Sericitization has increased the porosity, and thus the permeability to surface waters, along the fractures and in large areas throughout the entire mass of the rock.

(c) Erosion has taken place more rapidly than oxidation because, as the canyons have been cut on each side of the mountain, rather steep soil-covered slopes have been formed; the impermeability of these slopes has caused a large part of the surface water to run down the mountain side, or soak into the soil for only a short distance; thus only the water from heavy precipitations—rain water or that from thawing snow—has passed down through the rocks. Oxidation has been slow, and only a comparatively thin blanket of oxidized leached capping covers the orebody. Variation in the thickness of the capping is governed by the topography of the mountain and the permeability of the rock.

(d) In recent geologic periods, at least, the elevation of the disseminated ores has been high—at present about 7,300 ft.; therefore cold winters and warm summers have doubtless governed to a certain degree the rate of oxidation.

3. *The Enrichment*

The occurrence of the two principal secondary copper minerals divides the enrichment of the disseminated ores into two different types: (1) where covellite is more abundant than chalcocite; (2) where chalcocite is more abundant than covellite. Although the ores of these two types blend into each other they are still distinct enough to be taken up under separate headings.

The greatest depth to which the enrichment extends is about 800 ft. below the surface, but the average thickness of the entire known orebody is 445 ft., and it is covered by an average thickness of capping of 114 ft.

4. The Covellite Enrichment

The ore in which covellite is the most important secondary mineral is located below the top of the mountain and upper levels. A rough boundary might be taken to include the surface and underground ore above Sub-level 5 of the underground mine, as shown on the longitudinal section, Fig. 1.

In the summer of 1914 the writer spent three months working on the R level. Observations made here will hold good for the levels from the top of the mountain to a number of levels below R.

The steam shovels and drills along some of these levels are working part of the time in ore and part of the time in capping; thus a good opportunity is afforded for studying both.

(a) *The Capping above the Covellite Ores.*—The capping is usually composed of several feet of soil and 50 to 100 ft. of leached porphyry. The leached rock is colored with the characteristic browns of the iron ores. Near the soil the brown color is lighter, but passing downward it shades into a darker brown until, in a transition zone of a few feet, the rock assumes the dark-gray color of the ore. When the bank is freshly broken by a blast these colors are most noticeable; after a few days the moisture dries out, and the brown fades, but the gray appears wet, due to the presence of sulphuric acid, together with copper and iron salts. A short time after this surface is exposed a greenish stain appears on the gray rock, extending from 5 to 20 ft. below the brown, and in the course of a few days the evaporation of moisture causes a thin coating of copper sulphate crystals to form.

On the R level a churn drill was used to drill 6-in. holes for blasting purposes. The holes were drilled along the edge of the bank, 40 ft. apart, and to an average depth of 45 ft. While drilling in the brown capping the bit was scoured and clean, but when working near the contact of the ore green spots appeared on it, and in the ore just below the surface or brown capping it was often covered with a visible coating of copper.

The latter is remarkable because while the drill was in operation the scouring action of the grit in the slime would prevent the formation of this coating; therefore it must have been formed between the stopping and the hoisting of the bit, which at most was not more than 4 or 5 min., thus indicating a fairly concentrated solution of copper sulphate.¹¹

In general, the observations point to the fact that during the warm dry months of the summer, when the rock is moistened only by an occasional shower, oxidation is rapid and the iron and copper sulphates,

¹¹ It must be understood that this rock was only slightly moist, but while drilling enough water was poured into the hole to make a fluid slime and the copper sulphate was dissolved in this water.

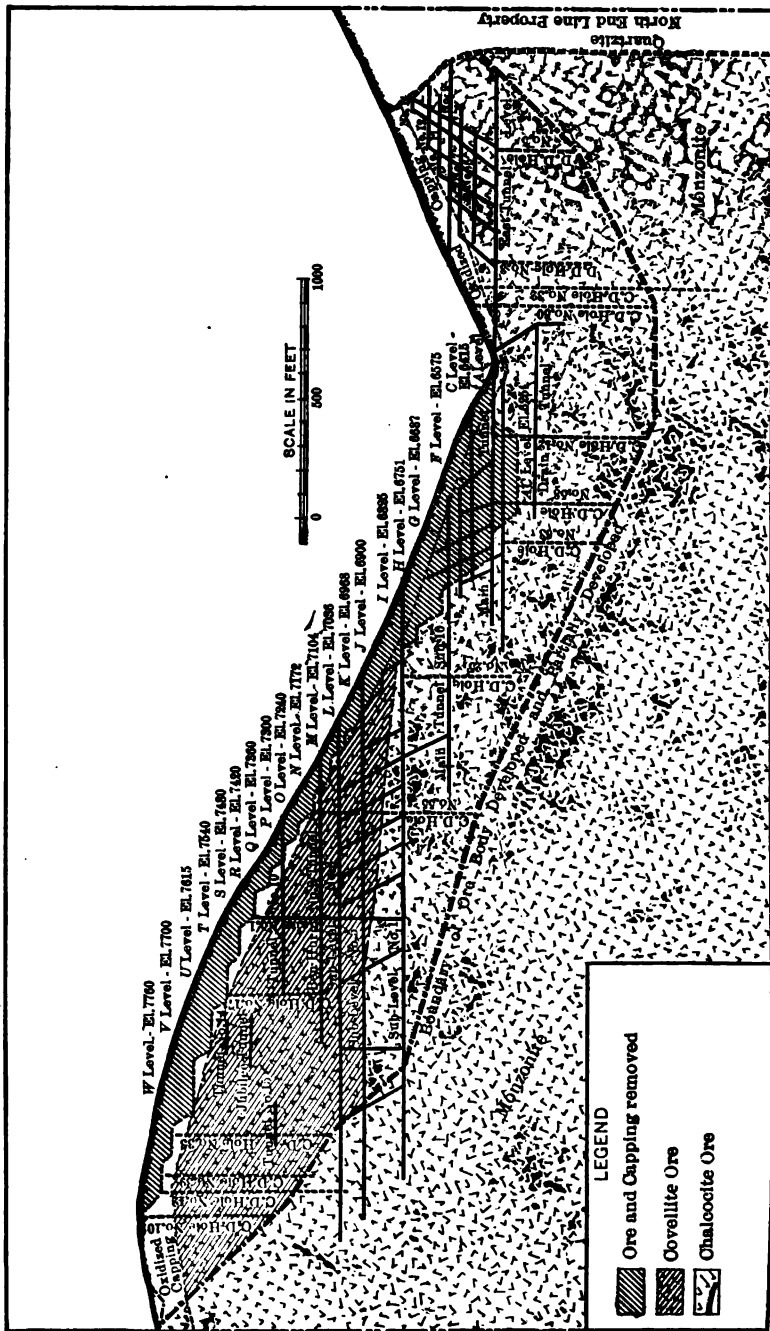


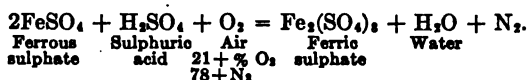
Fig. 1.—LONGITUDINAL SECTION THROUGH CENTRAL PORTION OF OREBODY OF THE UTAH COPPER CO., BINGHAM CANYON, UTAH.

together with sulphuric acid, accumulate in considerable amounts just below the capping.

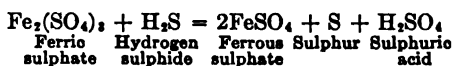
The water from the first heavy precipitation, percolating through the rocks, dissolves the accumulated salts, producing a more concentrated solution than those which follow for some time. In regions similar to Bingham, where the snow lies on the mountains for a number of months during the winter, oxidation may be slow during the cold weather and thus the solutions which pass down through the rocks will be almost barren of copper and iron salts—possibly until about the same time in the next year. In this case it would be remarkable if the periodic solutions were always of uniform acidity and always carried the same amounts of copper and iron sulphates in solution.

If the secondary minerals showed no signs of periodic changes little would be gained by discussing conditions which would produce changes in the descending solutions, but the disseminated ores do show these changes to a marked degree, as will be shown under the heading "Secondary Chalcopyrite and Bornite."

Another interesting phenomenon which was noted in connection with the churn-drill holes was as follows: In the brown capping a candle burned brightly while being lowered to the bottom of the hole, but in the ore the flame was extinguished a few feet below the surface, or the brown capping. Running the bailer up and down in the hole did not mix the gas and air in such amounts that the condition was improved. No odor of sulphur gases could be detected. It is therefore probable that the oxygen of the air was being rapidly used up, probably by the ferrous sulphate thus:



From this it would appear that the ferric sulphate had been reduced to ferrous sulphate a short distance below the capping. Whether this be so or not this phenomenon would indicate reducing rather than oxidizing conditions. It is usually supposed that the ferric sulphate may extend some distance below the oxidized zone, but if the sulphuric acid does become concentrated just below the brown capping, its tendency to produce hydrogen sulphide by reactions with certain sulphides would be increased. The hydrogen sulphide thus generated would rapidly reduce the ferric to ferrous sulphate thus:¹²



An excess of this hydrogen sulphide under low pressure might precipi-

¹² W. H. Emmons: The Enrichment of Sulphide Ores, *Bulletin No. 529, U. S. Geological Survey*, p. 91 (1913).

tate both iron and copper sulphides. The remaining hydrogen sulphide, together with that which may be generated at a greater depth, passes downward with the sulphate solution, carrying with it small amounts of colloidal copper and iron sulphides. Sphalerite is one of the sulphides which produce hydrogen sulphide most readily by the action of sulphuric acid, but it has been found experimentally¹³ that under a pressure of two atmospheres the reaction ceases or becomes extremely slow; thus this reaction would take place near the surface, but at comparatively shallow depth the sphalerite may remain unchanged in the presence of acid solutions. Other sulphides will build up greater pressures, but in general the generation of hydrogen sulphide by descending acidic surface solutions is probably a reaction which takes place a short distance below the surface in fairly concentrated acid solutions.

The minerals present in the capping are shown in thin section to be a large amount of secondary quartz which the surface waters have not affected; sericite is partly changed to kaolinite, and often the sericite and kaolinite fill the space once occupied by the original feldspar crystals. In some slides biotite has completely disappeared, leaving only small yellow crystals of rutile in its crystal outline; in other slides the biotite is bleached until it has the appearance of muscovite. The sericitization and bleaching of the biotite is largely a hydrothermal alteration, but the kaolinization is undoubtedly an alteration produced by acidic surface waters. The carbonates, azurite and malachite, are noticeably absent along the upper levels. The sulphides when present in the capping are often surrounded by the brown and yellow oxides of iron. One occurrence of gypsum was noted on the R level, but this was confined to small veinlets in a clay-like gangue about 30 ft. thick.

Aside from the kaolinite and small amount of gypsum there are no alteration products which would indicate a considerable neutralization of the acid solutions by the silicate minerals. It is therefore reasonable to suppose that the copper minerals in and just below the capping will show the effects of acid solutions to a marked degree while the sulphides at depth will show these effects to less degree.

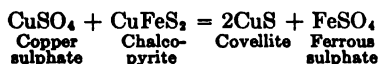
Polished sections made from ores, taken from the upper levels just below the capping, show the disseminated sulphide grains to be coated with covellite. The coating of covellite is usually much deeper on the chalcopyrite grains than on the pyrite; and when it is found as a coating, partly or completely encircling the pyrite grains, it is usually possible to see fragments of chalcopyrite in the covellite, indicating that the covellite was formed by the replacement of chalcopyrite rather than pyrite.

A, B, C, and D, Plate IV, are photographs of polished sections showing small veinlets, full width, together with the enlargements of small areas

¹³ Laboratory of Physical Chemistry, Stanford University.

of each vein. These veinlets were all collected along the R level just below the brown capping. In these sections, as in others from the same vicinity, covellite is shown to be the predominating secondary copper sulphide.

The formation of covellite from chalcopyrite—*A*, Plate IV—may have been quite simple, as shown in the following equation:



But in the replacement of galena—*D*, Plate IV—the reaction was probably more complex and could not be indicated by a simple equation because free sulphur was among the other products. The replacement of enargite by covellite would also represent a more or less complex replacement.

In the hand specimen a small veinlet may appear to be composed of pure chalcocite, but a polished section often shows this veinlet to be composed almost wholly of pyrite with microscopic veinlets of chalcocite passing through the pyrite; an example of this is shown in *B'* and *B*, Plate IV. *B'* shows the vein composed of pyrite while *B* shows the microscopic veinlets of chalcocite.

b. Conclusions Drawn from a Study of the Capping and the Covellite Ores.—There is a transition zone of only a few feet, between the oxidized brown capping, carrying practically no sulphides, and the gray rock containing large amounts of sulphides.

In the brown capping oxidizing conditions exist, while in the gray rock just below it reducing conditions predominate.

The copper and iron sulphides, together with sulphuric acid, accumulate just below the capping during the warm summer months, and are dissolved by the water from the first heavy precipitation after the dry season, producing a solution with a higher concentration of these salts than the solutions which follow for some time.

The solutions producing enrichment are periodic. The relative concentrations of dissolved salts and acid vary in different periods.

The solutions passing downward through the orebody are probably slightly acid to a considerable depth unless neutralized by the reactions with sulphide minerals.

Covellite is the predominating secondary copper mineral, formed under slightly acid conditions, in the porphyry below the upper levels.

c. Covellite Enrichment at Depth.—From the underground mine located directly below the top of the mountain and the upper levels, which have just been described, the writer has collected about 125 samples. These samples were taken on the three main levels, Sub 1, Sub 5, and Sub 8, at varying intervals along the crosscuts, perpendicular to the plane of the longitudinal section (Fig. 1). These crosscuts extend from the quartzite

on one side, directly across the intrusive nearly to the surface of the mountain on the other side, a total distance of about 2,100 ft. on Sub 8, and a somewhat greater distance on the levels below.

The polished sections of these ores invariably show that the enrichment has been similar to that of the disseminated ores of Ely, Nev., described by Mr. Spencer¹⁴ in the following words:

"Both chalcopyrite and pyrite have been replaced by chalcocite but the coatings of this secondary mineral are commonly much deeper on the chalcopyrite than on pyrite, much of which has not been coated at all."

Covellite, however, is more abundant than chalcocite in the ores of the underground mine and is formed in these ores chiefly from chalcopyrite by slightly acid or neutral solutions carrying a considerable amount of copper and ferrous sulphates. Covellite is often replaced by chalcocite, and in some sections it appears that the reverse order may have taken place.

Typical examples of the ores from the underground mine are shown in A, B, C, and D, Plate V. These illustrations also show the replacements which have just been described. In A and B it will be noted that there are small lath-shaped crystals of sericite penetrating the sulphide grains. These sericite crystals are invariably surrounded by covellite, chalcocite, or the secondary mineral, showing that they are not inclusions but have penetrated the grains from the outside and are therefore later than the primary and earlier than the secondary sulphides. The sericite crystals, although microscopic in size, have formed sub-capillary channels into the sulphide grains, thus permitting a more complete replacement by the secondary minerals. These examples also emphasize the importance of sericite as a mineral which increases the porosity of the rock.

D shows bornite and chalcocite forming the center of a grain, while covellite appears to have replaced part of the chalcocite; then a change in the solutions has caused a part of the covellite to be replaced by a border of chalcocite.

Chalcocite is formed more commonly from bornite—C, Plate VI—than from chalcopyrite, but under certain conditions the blue chalcocite may develop directly from the chalcopyrite and in a few small fissure veins white chalcocite may replace sphalerite—D, Plate VI. In general, there are probably two factors which govern the formation of chalcocite in the disseminated ores: (1) decreasing acidity in the solutions, probably to the extent of neutral or alkaline conditions; (2) relative concentration of copper sulphate to ferrous sulphate; the latter will be discussed under the heading "Secondary Chalcopyrite and Bornite."

¹⁴ Arthur C. Spencer: Chalcocite Enrichment, *Economic Geology*, vol. viii, No. 7, p. 622 (Oct., 1913).

5. *The Chalcocite Enrichment.*

The ores in which chalcocite is most prominent include those of the Pit and a number of levels above it.

While the capping in this area has many minerals in common with the capping which has been described, the most noticeable difference is the presence of the green and blue carbonate stains in the capping and gray ore.

a. The Carbonates in the Capping above the Chalcocite Ores.—The carbonates, together with the oxides and sulphides of copper, are present in the capping in such amounts that this capping is being deposited in large dumps where it will be treated in the future by leaching processes.

The formation of the carbonate ores may have been effected by a number of different processes, among which the most probable are: (1) The direct action of the copper sulphate solutions with lime carbonates in the rock. But the thin sections have failed to show the presence of calcite or gypsum, either or both of which should be present if this action had taken place. (2) The water running down the canyon from the limestone ridges higher up carries small amounts of $\text{H}_2\text{Ca}(\text{CO}_3)_2$ (acid calcium carbonate), which might produce carbonates, but again gypsum should be present in the ores. (3) The presence of carbon dioxide—both of atmospheric and organic origin—in the surface waters, in sufficient concentration, will produce copper carbonates when these waters come in contact with copper sulphate and sulphides. The geologic evidence shows that the latter has been the most effective process in the formation of the carbonate ores in the disseminated deposit.

Before steam-shovel operations began in 1906 the orebody was a short distance below the surface of a brush-covered mountain. It appears that during the summer months the decay of organic matter produced carbon dioxide, part of which remained in the soil; this carbon dioxide, together with atmospheric CO_2 , was evidently dissolved in the water which passed down the mountain side. At the top, the carbon dioxide content was probably small, but as the water passed down the mountain side the concentration of this gas and alkaline carbonates evidently increased until, about half way down, the concentration was sufficient to produce carbonates when these waters came in contact with the accumulated copper sulphate. The atmospheric oxygen also present in these solutions was probably used up in the oxidation of the copper and iron sulphides and also in the formation of melaconite and cuprite. The sulphuric acid formed in the zone of oxidation would doubtless be partly neutralized by the surface waters. Thus the presence of the carbonates indicates that the solutions were probably slightly acid to alkaline.

b. The Formation of Chalcocite.—It is safe to say that the chalcocite in the ores below this area was formed by weakly acid to slightly alkaline

solutions. *A*, Plate VII, shows the replacement of chalcopyrite by chalcocite, with bornite as an intermediate product. Later, the chalcocite has been replaced by melaconite and malachite. The melaconite may or may not be an intermediate product between chalcocite and malachite. In *D* it appears that chalcopyrite has been partly replaced by covellite; then a change in the solutions has caused the chalcocite to replace in part both covellite and chalcopyrite, leaving the covellite as residual crystals in the chalcocite. *C* shows chalcocite replacing pyrite, and a later replacement of chalcocite by melaconite; this replacement of pyrite by chalcocite may be a deep-seated change brought to the surface by erosion. In this specimen crystals of chalcocite shown by etching are at least $\frac{3}{8}$ in. long and are formed by replacement of bornite, chalcopyrite, and pyrite by the copper in the descending solutions. Etching with nitric acid or potassium cyanide develops a structure almost identical with that which has been attributed to primary chalcocite; it is therefore doubtful whether etching tests can be used as criteria to distinguish primary and secondary chalcocite.

In general, the chalcocite is formed in these ores by the replacement of chalcopyrite, bornite, pyrite, and occasionally covellite. The chalcocite usually forms directly from these minerals without intermediate copper-iron or copper sulphides; however, under certain conditions chalcopyrite may be an intermediate product between pyrite and chalcocite; also bornite may be an intermediate product between chalcopyrite and chalcocite; the latter is not common, and in the same specimen the writer has not observed pyrite passing through the intermediate stages of chalcopyrite, bornite, and covellite to chalcocite, as the theoretical replacement should proceed.

In the disseminated sulphide grains the replacement by chalcocite may vary from a thin coating to complete replacement, but here, as is common throughout the orebody, the enrichment as a whole is very regular.

Whether or not the intermediate products will be formed in the replacement of iron or copper-iron sulphides by chalcocite is doubtless governed by the concentration of the ferrous sulphate in the solutions producing the reactions.

6. *Secondary Chalcopyrite and Bornite*

Secondary chalcopyrite may be formed as an intermediate product between pyrite and chalcocite; and bornite as an intermediate mineral between chalcopyrite and chalcocite, as has already been discussed, but the most interesting occurrence of these minerals is where they are formed in the reverse order by the replacement of chalcocite and covellite.

In different parts of the disseminated chalcocite and covellite ores, especially in the deeper ores, the sulphide grains appear, in the hand

specimen, to be composed wholly of chalcopyrite; but in the polished section the chalcopyrite is often shown to be only an extremely thin coating surrounding the grain. It was probably these ores which Mr. Emmons referred to in the following quotation:¹⁵

"In much of the ore, especially in that of lower levels, chalcopyrite is an important ore mineral, and considerable masses . . . carry very little chalcocite. It is not known whether the chalcopyrite is a primary or secondary sulphide."

In the masses referred to as carrying very little chalcocite, it is probable that there was considerable chalcocite present, but it was covered by a chalcopyrite coating.

There are unquestionably two generations of chalcopyrite: (1) the primary, which is the most important ore mineral in the disseminated ores; (2) the secondary, which composes a minor part of the chalcopyrite. In *D*, Plate VIII, the primary and secondary chalcopyrite are shown. The primary chalcopyrite, marked *P*, forms a center surrounded by chalcocite and bornite and also one end of the grain, while the secondary forms a thin coating which in the photograph is shown as a border surrounding the grain.

The photographs in Plate VIII show the different stages in the replacement of chalcocite by chalcopyrite. In *A*, *B*, and *C* it appears that the white chalcocite has changed to a pale-blue chalcocite, due to a slight addition of iron, leaving the white chalcocite only as narrow gashes along crystallographic directions in the sulphide grain. By a gradual addition of iron the impure blue chalcocite is changed to bornite, and a further addition of iron and bornite is replaced by chalcopyrite in the form of a thin coating completely surrounding the grain, and also as small fissures and gashes extending into the grain. In *A*, *B*, and *C* the bornite and chalcopyrite are undoubtedly later than the blue and white chalcocite; the only question is the relation of the blue and white chalcocite. In the chalcocite ores which do not show the reverse reaction the white chalcocite is often formed by the replacement of the blue and it is possible that the white chalcocite in *A*, *B*, and *C* was formed by a replacement of originally blue chalcocite and is thus later; but if this has taken place there has also been a later replacement of part of the white by the blue chalcocite.

Experimental work¹⁶ tends to show that the white chalcocite is the purest form of chalcocite, while the blue is an impure variety containing a small amount of iron; thus it might easily be formed as an intermediate product in ordinary replacements of chalcocite or in the reverse reaction where chalcocite is replaced by bornite and chalcopyrite.

¹⁵ W. H. Emmons: The Enrichment of Sulphide Ores, *Bulletin No. 529, U. S. Geological Survey*, p. 179 (1913).

¹⁶ Laboratory of Physical Chemistry, Stanford University.

The photographs *A* and *B*, Plate VI, and *A*, *B*, *C*, *D*, *E*, and *F*, Plate VIII, were selected from specimens in which all of the sulphide grains showed the reverse reactions, so these illustrations show characteristic replacements common to the specimens, rather than selected grains picked to prove a certain point.

In some of the sulphide grains it appears that a border of chalcopyrite and bornite is formed; the chalcocite assumes the pale blue color; then the chalcopyrite appears to build from the center of the grain toward the outside and from the outside toward the center. In this way chalcocite and chalcopyrite with bornite often occur as very irregular patches in the grain, but in no case has the typical graphic intergrowth of chalcocite and bornite been observed. However, the irregular formation of chalcocite with bornite and chalcopyrite often approaches a structure almost as irregular as that of the graphic intergrowth formed by eutectic mixtures of chalcocite and bornite as described by Laney¹⁷ in the Virgilina ores.

In some grains this occurrence of chalcocite, bornite, and chalcopyrite could be explained by an incipient breakdown of the bornite into chalcocite and chalcopyrite, as described by Graton and Murdoch,¹⁸ if it were not for the fact that in some grains (*A*, Plate VIII) the ratio of chalcocite to chalcopyrite is too large, and in other grains (*F*) the ratio of chalcopyrite to chalcocite represents a larger amount of chalcopyrite than could be accounted for by iron in bornite. Other sulphide grains show the chalcopyrite as gashes in, and bordering, sulphide grains, the centers of which are composed wholly of bornite.

Some of the sulphide grains suggest that the chalcocite in the center of the grain might have been formed by the replacement of bornite, but in grains where bornite containing gashes of chalcopyrite is being replaced by chalcocite—*C*, Plate VI—the replacement proceeds from the outside toward the center. The bornite is more readily replaced than the chalcopyrite and thus the gashes of chalcopyrite are seen extending from the bornite into the chalcocite, and often are not replaced even after the complete replacement of bornite. In the grains showing the reverse reactions this selective replacement does not occur. *C*, Plate VI, might be taken to show the appearance of a sulphide grain similar to *E*, Plate VIII, after chalcocite-forming solutions had completely replaced the chalcopyrite border and partly replaced the whole grain.

In studying the specimen from which the photographs of Plate VIII were taken, the writer has observed, in one or two grains, two generations of chalcocite, but usually it occurs in such a way as to show that there has

¹⁷ F. B. Laney: The Relation of Bornite and Chalcocite in the Copper Ores of the Virgilina District of North Carolina and Virginia, *Economic Geology*, vol. vi, No. 4, pp. 339 to 411 (June, 1911).

¹⁸ L. C. Graton and Joseph Murdoch: The Sulphide Ores of Copper. Some Results of Microscopic Study. *Trans.*, xlv, 49 (1913).

been a gradual migration of the iron from the chalcopyrite border veinlets and gashes. This strongly suggests that a part of the blue chalcocite and bornite is formed as in the chalcopyrite by an addition of iron from the solutions acting on the outside of the grain.

It is interesting to find that in the covellite ores the reverse reaction also takes place and the covellite appears to be replaced directly by chalcopyrite. In some of the sulphide grains a small amount of secondary bornite is formed, but this bornite does not appear to be an intermediate mineral between the covellite and chalcopyrite. A grain of sulphide composed almost wholly of covellite is often covered by a thin coating of chalcopyrite; A and B, Plate VI, show this replacement. In A there is an indication of two distinct borders of chalcopyrite—marked I and II—separated by a narrow strip of covellite, which indicates that there have probably been two generations of secondary covellite and chalcopyrite. In both the covellite and chalcocite ores the chalcopyrite is formed by an addition of iron, the larger part of which has unquestionably been supplied by solutions acting on the outside of the sulphide grains.

The author believes that the secondary chalcopyrite has been formed by descending surface waters rather than the primary solutions, because chalcocite grains coated with chalcopyrite are often found near the top of the sulphide ores where enrichment extends 400 to 500 ft. below. If the chalcocite and chalcopyrite coating had both been formed by primary solutions, the thin chalcopyrite coating would undoubtedly have been replaced by secondary chalcocite or covellite. The only conclusion which can be drawn is that the local concentration of iron in the descending solutions has been responsible for the reverse reactions.

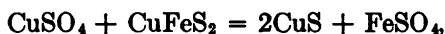
The formation of chalcopyrite by cold descending solutions has often been questioned, but recent work in the Laboratory of Physical Chemistry, Stanford University, has shown that chalcopyrite can be formed in cold solutions by the action of ferrous sulphate on chalcocite, covellite, or bornite. The data in the following paragraph, together with the photographs appearing in Plate IX, are published by the courtesy of Prof. S. W. Young.

Experimental work shows that chalcopyrite can form by the direct replacement of copper in chalcocite, covellite, or bornite, by the iron of the ferrous sulphate. The reactions take place in acid, neutral, or slightly alkaline solutions in the presence of hydrogen sulphide under pressure, but are more rapid in neutral than in acid or alkaline solutions.

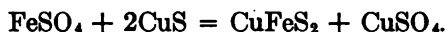
The photographs in Plate IV show synthetic chalcopyrite as borders and small veinlets covering and passing through fragments of chalcocite, bornite, and covellite which have been subjected to the action of ferrous sulphate for a period of about six weeks at a temperature of 30° C. A visible coating of chalcopyrite is formed on small fragments of chalcocite, covellite, or bornite in about three days; longer periods show an increase

in the intensity of the chalcopyrite color, and polished sections made from these fragments show, under the microscope, unquestionable chalcopyrite borders and small veinlets passing through the sulphide fragment. *B* shows quartz fragments in bornite, the contact completely surrounded by narrow borders of chalcopyrite. Future work will probably show that bornite can be synthetized in a similar way.

Hydrogen sulphide probably plays the part of a catalytic agent, and its expansion from a liquid state in the sealed tube causes considerable pressure, which is probably a factor in the reverse reaction. The fact that the reaction takes place in acid, neutral, or alkaline solutions emphasizes the fact that the reactions are governed by the mass law; thus it appears that in the presence of certain concentrations of copper sulphate, covellite is formed from chalcopyrite thus:



but if the concentration of the ferrous sulphate is increased beyond a certain point the reaction takes place in the reverse direction, thus:



Similar equations might be written to show the reverse reaction with chalcocite and bornite.

Strong evidence supporting the reversibility of some of the equations, common to the chemistry of secondary enrichment, is given in the disseminated ores. Thus the alternating borders of covellite and chalcopyrite in the covellite ores, *A*, Plate VI, are easily explained by changes in the concentration of the iron and copper in the descending solutions. These changes might be caused in the following manner: Certain veins, pockets, or zones in an orebody are often rich in copper and others are composed almost wholly of pyrite. While the sulphides higher in copper are being oxidized the descending solutions will carry a considerable amount of copper sulphate and chalcocite, and covellite will be formed to a considerable depth; but when pyrite is undergoing oxidation the amount of ferrous sulphate will be larger and a part of the chalcocite and covellite at depth may be changed back into chalcopyrite. The change in the concentration of the iron and copper is often gradual and thus bornite replaces part of the chalcocite; then as the concentration of iron increases chalcopyrite is formed.

If the solutions are periodic, as indicated in the capping above the covellite ores, the concentration of copper sulphate and ferrous sulphate might vary considerably in different periods, and therefore it would be quite possible to have two or more generations of secondary copper and copper-iron sulphides. Where such changes take place the relation of the primary and secondary minerals will be very complex. It is very

probable that the concentration of the ferrous sulphate must be high before the order of the replacements is reversed.

Analyses of waters from copper mines usually show the ferrous sulphate in considerable excess of the copper sulphate. Of the 18 analyses given by Emmons¹⁹ the iron sulphate (ferric near the surface and ferrous sulphate at depth) is present in excess of the copper sulphate in all but two samples.

In any event to a certain depth the net result is enrichment, because either chalcocite or covellite is present and a considerable amount of the pyrite is replaced by chalcopyrite. C, Plate V, shows a pyrite grain partly replaced by chalcopyrite. The chalcopyrite appears to be formed by a replacement of chalcocite rather than by a direct replacement of the pyrite.

At present it is impossible to say to what extent ferrous sulphate has effected enrichment, but possibly some of the reverse reactions in the Butte ores, described by Sales²⁰ in the following quotation, were caused by ferrous sulphate rather than by primary solutions.

"Bornite . . . is often seen to be a transition product between pyrite and chalcocite; also in the reverse reaction where chalcocite passes by gradual alteration stages through bornite to chalcopyrite While bornite is commonly an alteration product, and therefore secondary, it probably does not often result from the action of downward moving surface waters, but is generally a product of ascending solutions. . . . The alteration of massive chalcocite to bornite is common, as is the change from enargite to chalcopyrite, and covellite to chalcopyrite."

Similar replacements have also been described by Graton and Murdoch,²¹ and Ray,²² but in general the primary solutions are favored. The author has not made an extensive study of the Butte ores, and as reverse reactions in them might be caused by the primary solutions as well as the secondary, the matter is open for discussion.

III. CONCLUSIONS AS TO THE FORMATION OF THE IMPORTANT SULPHIDES IN THE DISSEMINATED ORES

Pyrite, chalcopyrite, and probably a minor amount of bornite are the primary copper minerals. The primary sulphides do not occur in these ores as magmatic sulphides, but owe their present distribution to the magnetite which was a magmatic mineral.

¹⁹ W. H. Emmons: The Enrichment of Sulphide Ores, *Bulletin No. 529, U. S. Geological Survey*, pp. 60 to 61 (1913).

²⁰ Reno H. Sales: *Economic Geology*, vol. v, No. 7, p. 682 (Oct.-Nov., 1910).

²¹ L. C. Graton and Joseph Murdoch: The Sulphide Ores of Copper. Some Results of Microscopic Study. *Trans.*, xlv, pp. 44 and 50 (1913).

²² J. C. Ray: Paragenesis of the Ore Minerals in the Butte District, Montana. *Economic Geology*, vol. ix, No. 5, p. 477 (July, 1914).]

Covellite is the most abundant secondary copper mineral forming by a replacement of chalcopyrite in slightly acid to neutral solutions.

Chalcocite is the most important secondary copper mineral formed under slightly acid to alkaline conditions.

Secondary chalcopyrite may be an intermediate mineral between pyrite and chalcocite; and bornite may be formed as an intermediate product between chalcopyrite and chalcocite.

Secondary chalcopyrite and bornite may also be formed by the replacement of the copper in chalcocite or covellite by the iron of the ferrous sulphate.

The concentration of the ferrous sulphate in the descending solutions is probably an important factor in secondary enrichment.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Illumination of Mines

BY ROBERT P. BURROWS,* CLEVELAND, OHIO

(New York Meeting, February, 1916)

IN preparing this paper the object has been to set forth facts relating to illumination problems, which, judging from the results realized in the iron and steel and other industries somewhat similar to mining, will tend toward furthering safety, production, and contentment of employees, as well as economy of operation in mines. By applying the principles of illumination with the assistance of modern appliances, the full benefits in efficiency may be derived from improvements already made in other details of mine operation.

The lighting of a typical coal mine may be divided into four distinct parts: 1, The lighting of the buildings about the top; 2, the lighting of the working faces; 3, general illumination at the bottom; and 4, special applications of lighting.

The lighting of buildings about the top may be treated in the same manner as that of any other industrial plant, for we have a boiler room, an engine and generator room, a forge, a machine shop, and a hoist room. These can be well and efficiently lighted by the use of 100-watt tungsten-filament multiple lamps with proper reflectors so spaced and suspended that a power consumption of from $\frac{1}{4}$ watt per square foot in the boiler room to 1 watt per square foot in the machine shop is obtained. The methods that apply to this kind of lighting have been ably treated by a number of authors,¹ and for this reason a detailed discussion is unnecessary.

The lighting of the working faces is usually done by means of portable lamps, of which there are four types in use: The oil torch, the acetylene lamp, and the oil and the electric safety lamps. The different types have been fully described in numerous papers and articles and will not be covered here, although a few figures on the cost of operation will no doubt be

* Engineering Department, National Lamp Works of General Electric Co.

¹ *Modern Industrial Lighting*. Issued by National Electric Light Association. Industrial Lighting, *Bulletin* 20, Engineering Department, National Lamp Works of General Electric Co.

Numerous papers on the various phases of industrial lighting have appeared in the *Transactions of the Illuminating Engineering Society*.

of interest. In attempting to obtain cost figures, one is impressed with the fact that apparently very few such data have been obtained in this country. It would seem that such data would be of particular benefit at this time, with the advent of the electric safety lamp.

The oil torch is without question the cheapest source of light. The acetylene lamp, at a cost of 6c. to 10c. per lamp per week, gives far superior illumination, but the characteristics of this source of light as well as any other open-flame lamp will bear careful consideration in view of the ever-present desire for industrial efficiency and safety. It is the opinion of many that the greater percentage of disastrous explosions in this country have resulted from the use of open-flame lamps in the so-called non-gaseous mines. This question of safety, of course, merits serious consideration.

The oil safety lamp has a distinct advantage in that it gives an indication of the presence of gas. Its development marked one of the greatest advances in mine lighting, although in most cases at the present time it is not considered a guarantee against explosion when in the presence of gas. Figures obtained from foreign countries indicate the cost of using oil safety lamps is from 7c. to 9c. per lamp per week.²

The electric lamp gives a steady and readily directed light, free from gases, soot, and frequent outage. A large proportion of the generated light is directed on the working face. It is sometimes considered a disadvantage that the electric safety lamp does not give an indication of gas as does the oil safety lamp. The trend of opinion in England, however, is toward choosing a lamp for the light it gives and to use some other means for gas indication.³ There is no question that an electric lamp passing the tests of the U. S. Bureau of Mines will give more light on the working face than any of the three previous illuminants, because it has been scientifically designed with that end in view.

Foreign practice has shown that electric light costs from 12c. to 17c. per lamp per week. This cost is about twice that of the oil safety lamp. The light on the "face," however, is materially increased by the use of the electric lamp. One foreign electric-lamp manufacturer places the cost of electric light at $2\frac{1}{2}$ c. per lamp per shift.² This figure, though it seems low, can well be realized in this country with a large installation and proper care. In this connection, it is very necessary to have proper housing and proper attention for electric lamps—more so than with the oil safety lamps. It has been found in foreign practice that this care and

² William Maurice: Electric Handlamps for Collieries, *The Electrician*, May 12, 1911.

F. J. Turquand: Design and Maintenance of Miner's Electric Handlamp, *London Electric Review*, March 6, 1914.

³ *Transactions, Institution of Mining Engineers*, vol. xlix, part 1, pp. 53 to 61 (1914-1915).

attention is very little, if any, more expensive than the attention that is given to oil safety lamps, even though more expensive help is needed, because fewer men are required to care for the electric outfits. This country has been slow in taking up the electric lamp. It has been said that in Belgium alone there are 12,000 outfits in use. The excellent work done by the U. S. Bureau of Mines to obtain the highest efficiency for this new source of illumination has accomplished what years of competition among electric mine-lamp manufacturers could hardly have brought about.

The application of the principles of industrial illumination to the general lighting of mines must be made in the face of conditions difficult to overcome. In fact, all the conditions the illuminating engineer considers most

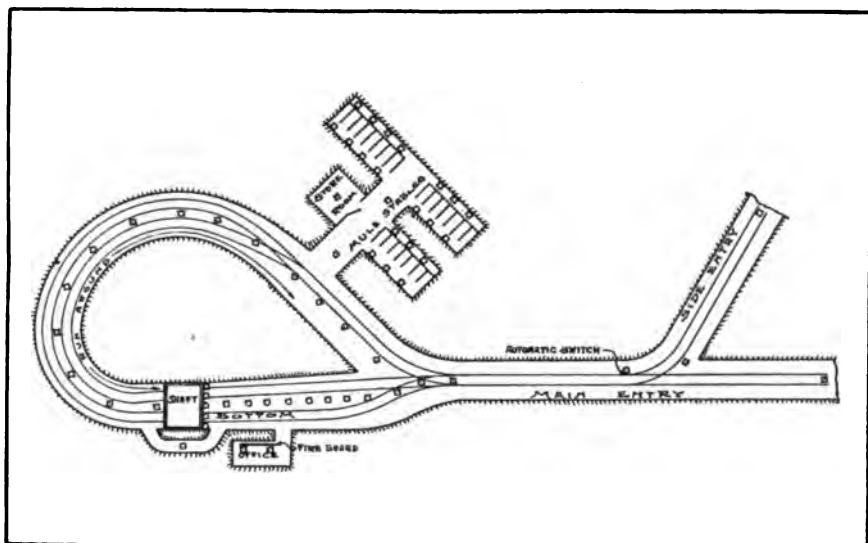


FIG. 1.—TYPICAL MINE LAYOUT.

difficult are present: low ceilings, black walls, dust, smoke, and dampness; but in spite of these, very satisfactory results have been obtained.

An ordinary mine, from a lighting standpoint, can be considered as composed of at least six parts: the bottom, the run-around, main entry, side entries, mule stables, and small rooms, such as offices, pump rooms, storage rooms, and first-aid rooms. These are shown diagrammatically in Fig. 1. The bottom, being the entry and exit for both men and coal, accommodates more traffic than any other part of the mine and should be especially considered from the standpoint of both convenience and safety. Fig. 2 shows a portion of a well-illuminated bottom and shaft opening of a typical mine. The lighting of the shaft in this case was accomplished by the use of 40-watt tungsten-filament lamps equipped with angle reflectors, placed above and across the shaft opening so as to direct the light on the

cages. The maximum intensity is at the near edge of the cage, and the eyes of the workmen on the side of the shaft toward the observer are not subjected to the glare of the lamps. For comparison, Fig. 3 shows this same portion of the mine lighted by the use of bare carbon lamps. It is



FIG. 2.—WELL-LIGHTED SHAFT BOTTOM.

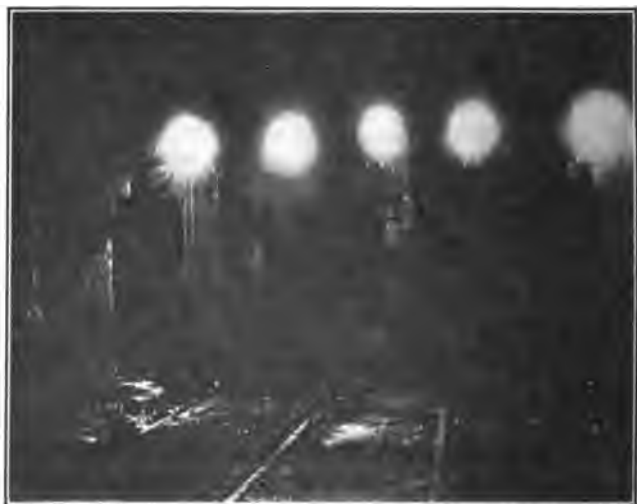


FIG. 3.—POORLY LIGHTED SHAFT BOTTOM.

readily seen that the distribution is not of the best and also that the glare of the bare lamps obscures that portion of the mine which lies beyond. These illustrations were made from actual photographs, retouched only enough to remove the halation effects of the bare lamps. The photo-

graph shown in Fig. 2 was exposed about 1 min. as against 15 min. for the one in Fig. 3.

| That portion of the bottom leading into the mine, where cars are directed on to the cages, can be well lighted with 40-watt tungsten lamps in shallow dome reflectors placed above and between the tracks. These units, spaced at about 6-ft. intervals and hung about 8 ft. above the floor, will give satisfactory distribution of light. It will be noticed from Fig. 4 that the car wheels are well illuminated and that there is practically no glare. It would be well to design the lighting of this part of the mine on a basis of 4 to 5 foot-candles at the floor, not because the work demands this intensity, but because of the greater safety which results from ample



FIG. 4.—PROPERLY ILLUMINATED ENTRY.

illumination and because dust collecting on the lamps and reflectors decreases the amount of light delivered.

The run-around should require only sufficient light to make visible any obstructions in the path of the empties as they leave the cages. This part of the mine may be illuminated with 25-watt tungsten lamps equipped with shallow dome reflectors, spaced 15 ft. apart and suspended 8 ft. above the floor. In the main entry, the function of light is not so much to illuminate as to silhouette objects which may obstruct the passageway. With silhouette lighting, a comparatively small amount of light is needed to obtain the effect desired, which is to see objects outlined against something that is lighted. For instance, whitewashed doors or walls reflecting the light toward the observer's eye are excellent backgrounds against which objects form silhouettes when in the line of vision of the observer. The glint of the light on the rails forms another good surface from which

silhouette lighting may be obtained. With 25-watt tungsten lamps in shallow dome reflectors, spaced at intervals of about 300 ft., the height depending upon the height of the entry, the silhouette lighting is excellent. Two units, one to illuminate the switch and the junction and the other illuminating a portion of both the main and side entries, help to eliminate collisions and by the increased light warn the trip driver that his train is approaching such a junction.

The mule stables with their low roofs may be effectively lighted with 40-watt tungsten lamps equipped with angle reflectors placed along the back wall and as high as possible, one unit to each two stalls. In front of the stalls and opposite the angle units, 25-watt tungsten lamps with deep bowl reflectors may be used to illuminate the feed boxes and passageway.

The mine offices need but one 25-watt tungsten lamp equipped with a shallow dome reflector. The fireboard at the bottom should be well illuminated with one or more 25-watt lamps of this type equipped with angle reflectors, depending upon the size of the board, while the pump rooms and storage rooms may be lighted in the same manner as offices.

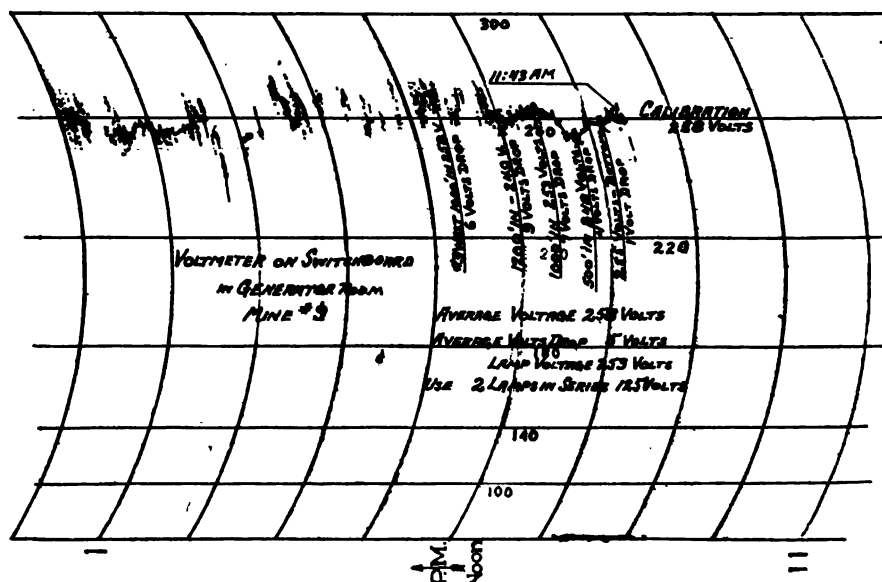


FIG. 5.—VOLTAGE RECORD OF MINE LIGHTING CIRCUIT.

The first-aid rooms, in order that the best attention be given the injured, should not only be well lighted, but should have the walls well white-washed, thereby obtaining well diffused and distributed light. Frequent whitewashing of the walls of the bottom, offices, mule stables, etc., and the walls of the entries for 20 ft. each side of the units, will greatly increase

the illumination in these parts of the mine. Carbon lamps are most generally used in mines, but to keep the load on the generator as low as possible and maintain the most constant illumination in spite of voltage fluctuation, and to direct the light where wall and ceiling reflection cannot be relied upon, tungsten-filament lamps with weather-proof enameled reflectors will, in my opinion, be found most satisfactory.

It may be interesting, by reason of the high voltage usually found in mines, and its fluctuation, to show how the proper voltage for a lamp, to secure greatest life and light, is determined. A recording voltmeter is connected at the switchboard on the terminals of the switch controlling the lighting circuit, usually the trolley line. When this is in operation, a carefully calibrated portable voltmeter is connected in multiple with the recording meter and a section of the chart of the recording meter is compared with the readings of the calibrated portable meter. Fig. 5 shows a section of such a chart and the calibrated line. This chart should be taken over a period of at least 3 hr. and for 24 hr. if possible.

TABLE I.—*Lamp Voltages for Various Line Voltages. Street-Railway Tungsten-Filament Lamps.*

Average Line Voltages	Number of Lamps in Series	Voltage of Individual Lamp
250	2	120
260	2	125
270	2	130
280	2	135*
290	2	140*
300	3	105
325	3	105
350	3	115
400	3	130
425	4	105
450	4	110
475	4	115
500	4	120

For voltages above 500, use five lamps in series as on street-railway circuits.

* Special lamps.

Voltage readings are then taken back from the shaft along the main entry at intervals of 300 or 400 ft. by means of the portable meter, the voltage and time being recorded. A study of the chart will show the average voltage over the period taken. A comparison of the chart with the voltage readings taken back in the mine will show the average drop in the line. From the average voltage obtained from the chart should be subtracted the average line drop obtained from the readings taken in the mine, the result being the voltage on which lamps will operate to give the same life as on the fluctuating voltage in the mine.

For average voltages up to 250 volts, regular multiple lamps should be used. For average voltages from 250 up to 280 volts, there is a choice of burning lamps in multiple or in series. The best practice is to burn two lamps, carefully selected for current, in series. Such lamps can readily be obtained and are known as street-railway lamps. For voltages above 280, the proper lamps should be selected for series burning. Table I lists lamps for specific voltages.

A recording chart, used as described previously, showed a maximum voltage variation of 20 per cent. From Fig. 6 it will be seen that with a 10 per cent. reduction in voltage, the candlepower of the carbon lamps is 20 per cent. lower than that of the tungsten lamps. Fig. 6 shows that the characteristics of the tungsten lamp are such that voltage variation does not affect candlepower as much as it does that of carbon lamps.

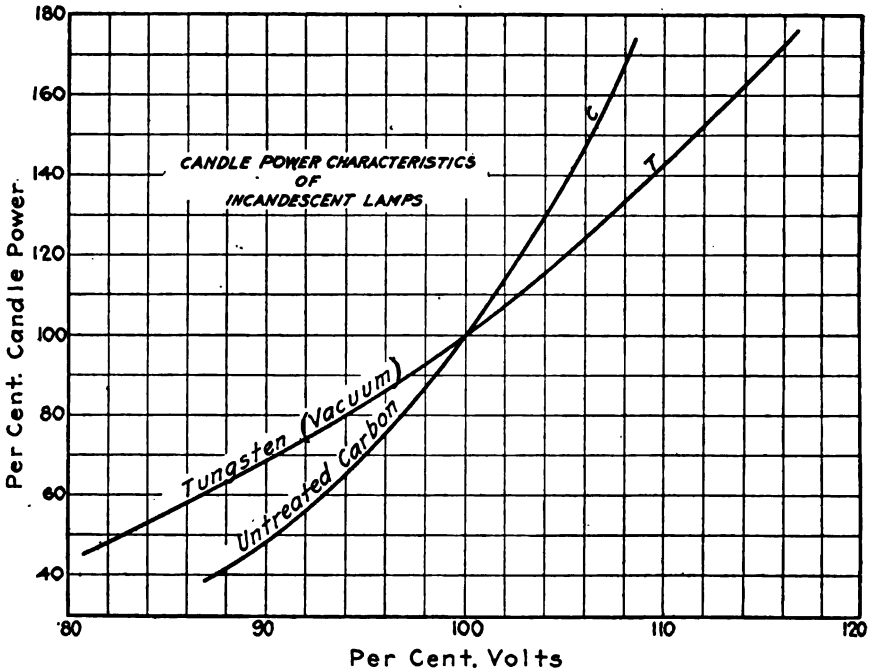


FIG. 6.—CANDLEPOWER CHARACTERISTICS OF INCANDESCENT LAMPS.

A few comparative cost figures in connection with the problem of more efficient illumination follow. Consider, for example, an installation, such as illustrated in Fig. 1, where twenty-six 40-watt tungsten lamps and reflectors and thirty-one 25-watt tungsten lamps and reflectors are to replace the same number of 32-cp. and 16-cp. carbon lamps, respectively. During a period of 300 days, at 10 hr. a day, the tungsten lamps would consume about 5,440 kw-hr., while the carbon lamps

would consume about 14,940 kw-hr. With the cost of current at 0.5c. per kilowatt-hour, the saving in cost of power with the use of tungsten lamps would be about \$50 a year. From this must be subtracted about \$17 for the difference between the cost of the carbon lamps and the tungsten lamps. This will leave about \$23 net saving. With the reflectors costing \$60, the installation would be paid for in three years.

These figures tend to show that if dollars and cents alone were considered, it would be more profitable to use the higher efficiency lamps. This is even more marked when the illumination on the working plane is considered, because with the use of reflectors the illumination is more than double that obtained with carbon lamps.

There are many other places where special applications of lighting would tend to increase efficiency and convenience; for instance, trip-lights—now as a rule simply oil torches on the end of the train—could be easily replaced by small storage-battery outfits showing a red light. Locomotive headlights can be equipped with low-voltage concentrated-filament tungsten lamps in parabolic reflectors, with a decrease in trouble, increased light, and decreased breakage over the present carbon or regular tungsten filament. Two 30-volt, 100-watt tungsten-filament locomotive-headlight lamps can be burned in series with a resistance. The loss in current through the resistance is a small factor as compared with the gain in steadiness and brilliancy of illumination from the parabolic headlights. The construction of this lamp is such that maximum strength of filament is obtained, which is an essential feature where the service is as severe as on a locomotive. Another possible consideration is the placing of distinctive lights where telephones are located, or where first-aid equipment may be obtained. This could be accomplished by the use of red lights on the power circuit installed in connection with a small primary-battery system, which would operate a miniature lamp in place of the large lamp should the power circuit for any reason fail. This system has been successfully worked out in theaters where the same principle is involved.

It is hoped that, from the few figures given in this paper, it will be seen that the application of the latest scientific knowledge to the lighting of mines is not so expensive as it is generally thought to be, and should be considered as a means of increasing safety, bettering working conditions, increasing production, and at the same time decreasing the cost of operation.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

The Evolution of Drilling Rigs

BY R. B. WOODWORTH,* PITTSBURGH, PA.

(New York Meeting, February, 1916)

INTRODUCTION

IN the sinking of bore holes, there are but two fundamental operations—drilling and hoisting,—which determine in the main the character of drilling mechanism and structures. There are endless ramifications, however, in the execution of these fundamental operations, according to the purposes for which the bore holes are drilled, their maintenance after completion, the protection and convenience of workmen, etc.

The three main lines of use for bore holes are in mineral exploration, the sinking of water and salt wells, and the exploitation of petroleum and natural gas. This paper has to do more especially with the last. Here again there are many divergences. Wells may be drilled by either the percussive, the hydraulic, or the abrasive method. Finally, each of these methods, as applied by drillers of diverse nationalities, has followed somewhat different lines of development. The American system of cable-tool drilling has perhaps had the widest application; the hydraulic rotary is also in extensive use; but the Canadian system, the Galician system, and the Russian free-fall system all have their points of recognized merit and are preferred by operators accustomed to their use. These differences in drilling procedure, which rest ultimately on essential variations in geological conditions, are reflected in the drilling structures as well as in the drilling mechanism.

The purpose of the present paper is to record those stages in the development of the application of steel to the construction of drilling mechanism and structures with which the writer has been intimately associated, and to contribute to the history of the art of drilling other data acquired in the course of his personal investigations. It is necessarily limited by reason of space to American practice, with especial reference, therefore, to the cable-tool system and the hydraulic rotary method of drilling wells for oil and gas.

* Engineer with Carnegie Steel Co.

Types of Drilling Structures

With these limitations in mind we may now proceed to discuss the line of development in specific types of drilling structures. It should be borne in mind, however, that in addition to the methods of drilling and hoisting there are variations in other features, due to the arrangement of engines, the depth and size of wells, the character of strata penetrated, the preservation and maintenance of wells after completion, the means taken for the convenience and protection of workmen, and for the prevention of fire, and likewise to the character of the service and to the removal and re-use of the equipment. Drilling structures are conveniently divided into drilling rigs and drilling machines. Drilling machines are mounted on wheels and are intended for transportation, by their own motive power or otherwise, from point to point without dismantling, and their essential feature is their relatively compact character and their portability as units. Drilling rigs are either intended to remain permanently at the location in which they are first placed, or may be removed from place to place only by resolution into their component parts by dismantling and by some means of locomotion other than their own power.

Portable drilling machines have been many years in use in a number of types. The operating parts have always been made of metal, and in the conversion of their frames and other structural parts from wood into steel the line of development has been exactly parallel with that of other machinery. These machines are equipped as a rule for shallow-well drilling only, though in a number of instances quite deep wells have been put down with them. This paper has to do with drilling rigs rather than drilling machines; hence the latter will receive incidental mention only, in the course of the discussion.

Again, we distinguish between a drilling rig and an outfit. The latter comprises not only the drilling structure but also the tools, engine, boiler, ropes and other accessories. In sinking oil and gas wells by contract, it is customary for the owner to furnish the drilling rig and the casing, while the tools, engine, boilers and other parts of the outfit are furnished by the contractor.

The arrangement of the various parts of a complete drilling rig, as used in the most modern deep-well drilling for oil and gas, is shown in Fig. 1, which exhibits diagrammatically the ground plans of the three types of drilling rigs in use in the United States together with their combinations.

With particular reference to oil and gas well practice, the line of evolution in drilling structures is summarily as follows:

1. The spring pole and tripod.
2. The braced mast, single or double.
3. The four-legged braced wooden derrick intended for permanent use, built without idea of removal.

4. The four-legged braced wooden derrick, put together with bolts to permit removal and re-erection.
5. The pipe derrick.
6. The long-panel structural steel derrick, built after the idea of what the designer thought the driller ought to use.
7. The short-panel structural steel derrick, built as nearly as possible along the lines of the braced wooden derrick.
8. The structural steel drilling rig complete, with machinery supports, wheels, house framing, metallic covering, etc.

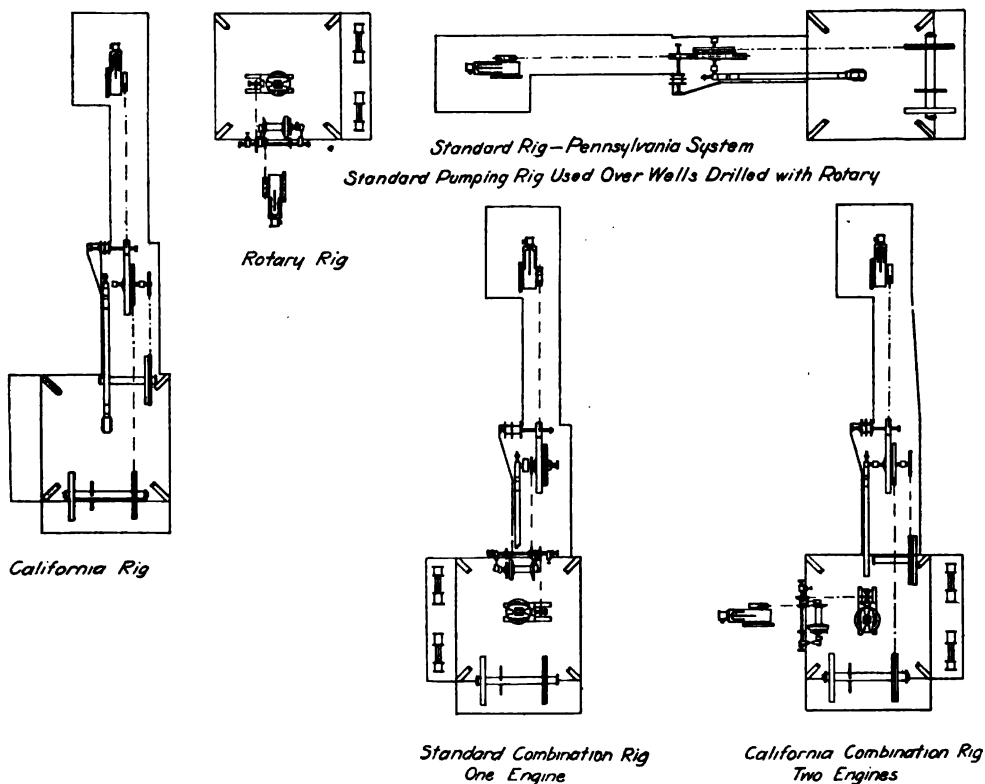


FIG. 1.—DRILLING STRUCTURES FOR OIL AND GAS WELLS. DIFFERENCES IN LOCATION OF OPERATING MECHANISM.

Our discussion will follow this division, based on the mast or derrick as the most outstanding and essential structural feature, but will also deal with incidental details of operating mechanism, so far as they are pertinent.

If in the treatment of steel derricks and drilling rigs the writer refers most largely to designs made under his supervision and executed by the company with which he is associated, it is because this company, while

not the pioneer in the manufacture of structural steel derricks in the United States, is today the only manufacturer in a persistent and systematic way. It is also true that the complete steel drilling rig was first designed by the writer and manufactured by that company and the development of the steel drilling rig in the United States is, therefore, a record of the improvements in manufacture introduced by that company under the supervision of the writer and his superiors and assistants in office and shops, with such further collaboration and assistance as he has been able to obtain from the officials of the Oil Well Supply Co. and like companies with whom in the first instance he co-operated. The reader will pardon, therefore, any note of commercialism there may be in statements intended solely as the record of a structural development.

I. SPRING POLE AND TRIPOD

The records of well drilling in the United States begin in 1806, when the first recorded artesian well put down in the United States was drilled for salt by David and Joseph Ruffner at the Salt or Buffalo Lick near Charleston, W. Va., in the Kanawha Valley, an account of which will be found in Vol. 1A of the West Virginia Geological Survey.

1. *Spring Pole and Tripod*

Their original supply of brine came from shallow pits and the flow was more or less weakened by the presence of surface water. It occurred to them to drill wells to rock, and thus obtain a purer, stronger supply, just as in later days it occurred to George H. Bissell to abandon the shallow pits from which oil was skimmed at Titusville and tap the source with a well. A study of these two events will serve to demonstrate the analogous character of the conditions. The salt wells were the pioneers of the oil and gas industry.

The appliances of David and Joseph Ruffner were most simple. Their walking beam was a spring pole mounted on a forked stick of wood. Their bits were $2\frac{1}{2}$ in. in diameter, quite primitive in construction, and attached to the end of the pole by a rope. Their tubing consisted of two long strips of wood whittled in half tubes and wrapped with twine. Their conductor was a straight well-formed hollow sycamore gum, 4 ft. in internal diameter, sunk to rock in a shallow pit. Yet the essential features of all subsequent development in the cable system of deep-well drilling were present. Jan. 15, 1808, on which day they drilled into the salt sand, is a record date in the art of drilling wells.

The first well drilled for oil was the Drake well, sunk by hand in 1859; but the next three or four wells drilled in the mad rush which followed were "kicked" down with the aid of spring poles, as were hundreds later

in shallow territory. This method, as described by J. J. McLaurin in *Sketches in Crude Oil*, second edition, 1898, afforded a means of development to men of limited means with heavy muscles and light purses. An elastic pole of ash or hickory 12 to 20 ft. long was fastened at one end to work over a forked stick or other fulcrum. To the other end stirrups were attached or a tilting platform was secured by which two or three men produced a jerky motion that drew down the pole, its elasticity pulling it back with sufficient force, when the men slackened their hold, to raise the tools a few inches. The principle resembled that of the treadle board of a sewing machine. Fig. 3, redrawn from McLaurin, p. 76, illustrates the essential features but, like many other similar sketches, it fails to show the hoisting equipment which was indispensable even at such wells.

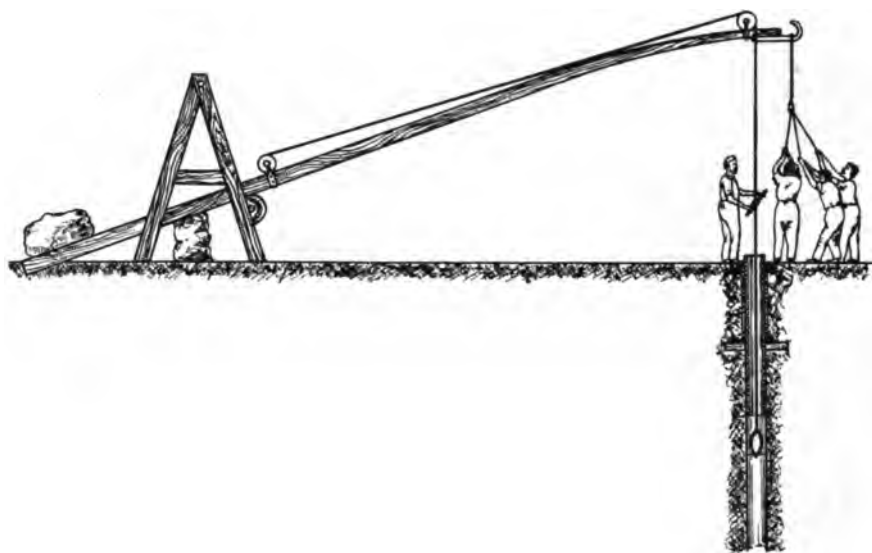


FIG. 2.—SPRING-POLE DRILLING METHOD USED BY JOBARD AT BRUSSELS, 1828.
(TECKLENBURG, VOL. IV, PL. II.)

It is true that the 3-in. sand pump referred to by McLaurin could be hoisted by hand from a considerable depth; but it is obvious that, with the exception of the most shallow wells, there must have been some means of hoisting provided. W. H. MacGarvey, of Vienna, intimately connected with the establishment of the petroleum industry in Canada, and one of the founders of the great Galician-Carpathian Petroleum Corporation, who has done perhaps more than any other single man to develop the Canadian and Galician methods of drilling, has given us an exact description in the preface to J. D. Henry's *Oil Fields of the Empire*. Fig. 4, taken from this work, shows in addition to the spring pole the lower portion of the tripod with the hoisting windlass. Mr. MacGarvey says:

"The early methods were exceedingly primitive, and at the present time many of us wonder how with such poor material such splendid pioneering results were secured. The first method of drilling in Oil Springs was by means of a spring pole. The process of putting down a well was slow and laborious, and many months were necessarily spent in drilling a well to a depth which at the present time would occupy two or three days. The first great Canadian spouter, the celebrated Shaw well, was drilled by this method."

According to publications dealing with this class of work and descriptive of operations in Canada, Ohio, New Zealand and elsewhere, the spring pole as a means of drilling oil wells disappeared about 1866. Its disappearance was due beyond question to the introduction of steam as a prime mover and the development of the portable drilling machine for the shallower wells. An excellent idea of the change and the reasons

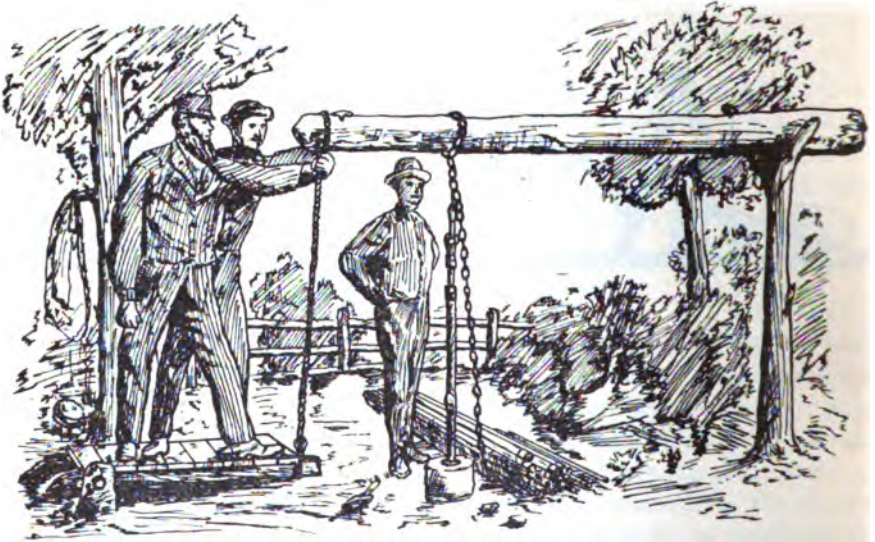


FIG. 3.—SPRING-POLE DRILLING METHOD. (McLAURIN, p. 76.)

therefor is given by the description of the early wells drilled in New Zealand contained in J. D. Henry's book entitled *Oil Fields of New Zealand*, 1911. It was seven years after Colonel Drake drilled the first well at Titusville that the New Zealand pioneers began their search for oil with the inferior spring-pole tools of that day. The first well was sunk by Messrs. Carter, Smith, Scott, and Ross, operating as Carter & Co., at the root of the breakwater of Moturoa. On Mar. 17, 1866, the men had reached the depth of 60 ft. by shaft, but the bottom of the shaft began to feel so shaky that they decided to continue their well by boring. Spring pole and tripod were taken down and a four-legged derrick erected; and boring began at the end of 1866. The derrick bore the placard "To Oil or London," and the well was put down to a depth of 300 ft.

The Peoples Petroleum Co. began the second or Beta well about 500 yd. from the Carter or Alpha. A 4-in. hole was sunk 516 ft. by spring pole and tripod, which bore the legend "To Oil or Edinburgh," and the well was named Victoria Well. Both wells were afterward taken over and deepened by the Taranaki Petroleum Co.

It may be noted in passing that these early wells were unsuccessful and that the first real oil produced in commercial quantities came from the No. 1 (Birthday) well of the Taranaki Petroleum Co., Ltd., completed on Sept. 24, 1906, to a depth of 2,345 ft. A steel derrick, said to be the only one then in New Zealand, was over this well at the time of Mr. Henry's visit in 1910. Early efforts to reach oil undoubtedly failed of success because of the inability of the operators to sink to the real oil-bearing strata.

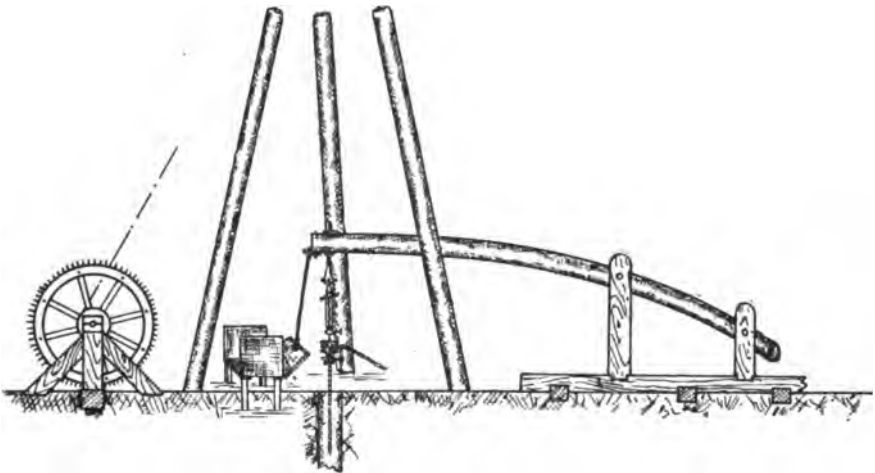


FIG. 4.—SPRING-POLE DRILLING METHOD. (W. H. MACGARVEY.)

It must not be imagined, however, that the principle of the spring pole based on its resiliency has disappeared from the art of drilling. In the fifth edition of a text-book on *Coal Mining* by Hubert W. Hughes, London, 1904, will be found on p. 17 the statement:

"In our coal districts the vibratory movement is often given to the rods by the use of the spring bar, which consists of a wooden pole having one end fixed to the ground, a fulcrum placed further on, and the rods attached to the other end. The blow is struck by depressing the beam, the rods being raised by the elasticity of it. The lengths of the parts on each side of the fulcrum are usually 1 to 3 or 5. For shallower holes the axis may be fixed, but for deeper ones it must be movable. An elaboration of this method consists in the employment of two spring poles. The first is from 60 to 70 ft. long fastened at one end and at two-thirds of its length from the fixed point it rests on an upright. To the other end are fixed two cross bars which the workmen press down on to a second spring pole, thus producing a dancing movement. Between the uprights and the cross beams is attached a hook, from which the boring tools are hung in the usual manner."

The first edition of this book appeared in 1892 and it may be that the practice referred to was current at that date; in any event, it was not considered obsolete when the fifth edition was printed.

Moreover, the principle is now employed in connection with the boring system exploited by John R. Thom, manufacturer of boring plants at

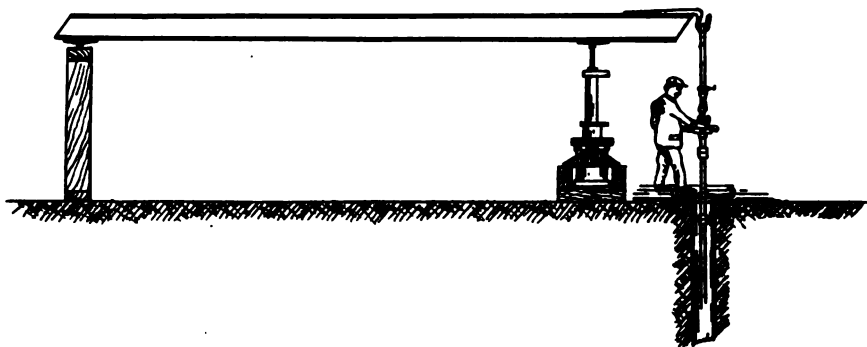


FIG. 5.—WALKING BEAM. THOM ENGLISH DRILLING SYSTEM.

Patricroft, England, as is shown by Fig. 5, re-drawn from a recent catalogue. The hoisting frame used in connection with this system is a tripod with windlass mounted across two of the legs, and a sheave hung from the top. This method of hoisting is identical with that still to be found in

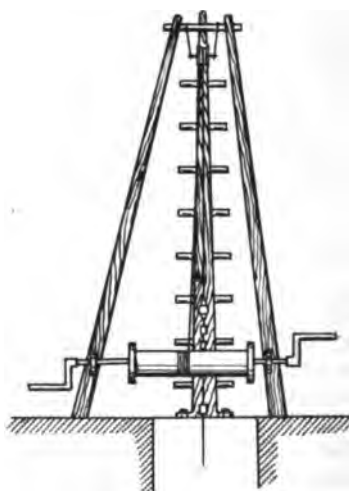


FIG. 6.—MINING SHAFT TRIPOD. (R. A. S. REDMAYNE.)

present mining practice, as shown in Fig. 6, taken from R. A. S. Redmayne's *Modern Practice in Mining*, Vol. I, London, 1908, p. 84, which is reproduced here as exhibiting the general resemblance of drilling methods whether employed in mineral exploration work or in the sinking of wells.

The Thom system has found quite a wide application. In Vol. XXIV of the *Petroleum Review* (1911), for example, will be found a reproduction of the structure used on Plot 233 of the Maikop Oil Proprietary Co., Ltd., and on Plot 65 of the Maikop Co-operative Petroleum Co., Ltd. In connection with the latter, it is to be noted that drilling was begun by hand, and that boring contractors in that field were surprised at the rapidity with which wells could be sunk by the Thom system.

2. Chinese Drilling Methods

It has long been known that the art of sinking wells for salt by drilling had reached a fair state of development in China by the year 1700, and that various travelers in that country had been much impressed with

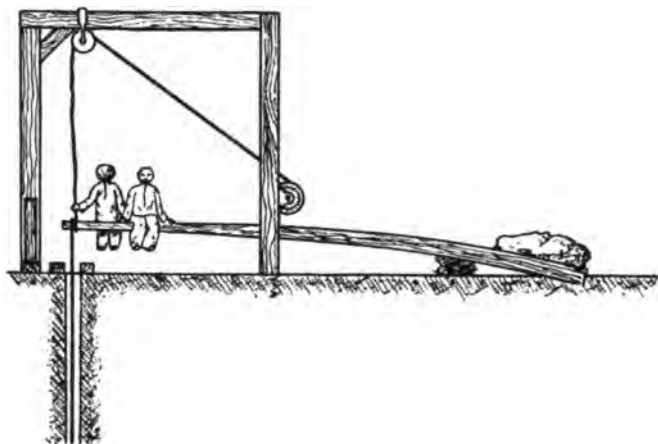


FIG. 7.—CHINESE DRILLING METHOD PER TECKLENBURG.

the ingenuity exercised in this art by the natives. Before the date named, Chinese in the district of Wu-Tung-Kiao had sunk over 10,000 wells to depths of 560 m., some even to 1,242 m. Von Richthofen narrates that the Chinese in the province of Ssu-Chan had drilled several thousand bore holes and had found salt strata at 600 m. and gas at 900 m.

Some rather fanciful descriptions have been published in the semi-technical journals. One is recalled, in which the Chinese coolies are depicted as running up over a spring board and jumping therefrom with great glee as if the business of drilling wells were recreation rather than sober industry. Fig. 7, taken from Tecklenburg's *Handbuch der Tiefbohrkunde*, Vol. IV, Plate II, is doubtless a more correct representation. It will be noted that the spring pole was counterweighted, and that the hoisting was done by a sheave suspended from a cross beam carried on a three-legged frame, the rope being wound around a windlass.

The most recent description of the petroleum, gas, and brine wells of

Ssu-Chan, based on an original article by Louis Coldre, appeared in the December (1914) number of *Western Engineering*, published at San Fran-

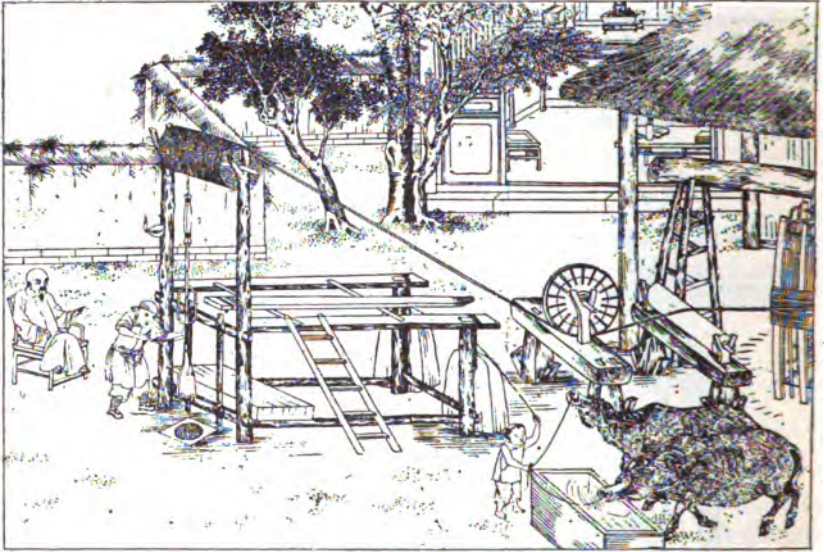


FIG. 8.—CHINESE DRILLING METHOD PER COLDRE. STARTING DRILLING.

cisco, and is well worth careful study. It indicates very great skill on the part of the Chinese. Fig. 8, taken from this paper by courtesy of the

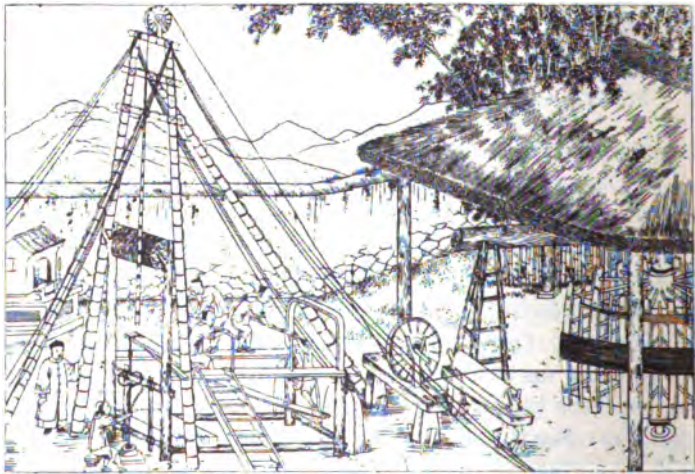


FIG. 9.—CHINESE DRILLING METHOD PER COLDRE. REGULAR DRILLING OPERATIONS.

editor, illustrates the methods of use when the wells are begun, and Fig. 9 the usual routine of regular well-drilling operations. When the well is

started, hoisting is done by a roller carried at the top of two forked poles. When the drilling has progressed to a considerable depth, additional poles are run up and braced to carry the crown sheave placed over the header beam at the top. The line is carried down to a vertical sheave, and thence to a horizontal drum; and the string of tools is suspended from the end of a lever made of a thick plank. This is so adjusted that the lever is horizontal when the bit just touches the bottom of the hole. The four workmen then jump on the plank, depressing it and raising the bit a couple of feet. When they jump back on the side benches, the bit falls. By jumping two at a time and passing from one side to another the men are able to drill at the rate of 12 to 15 strokes per minute. The double-tripod derrick, if it may be so called, is built of bamboo and guyed with hemp ropes. The equipment, while simple, embodies the necessary mechanical features and is structurally adequate.

3. *Present Use of Tripod*

As already noted, the tripod is in modern use in the Thom system of boring. Probably its widest use is in connection with mineral exploration work, and the characteristic feature of the modern exploration outfit as used, for instance, on the iron ranges of Minnesota, is a portable drilling machine with a hoisting equipment consisting of a three-legged frame supporting a sheave at the top and carrying a windlass. Such tripods are used in heights of 18 and 23 ft., for example, in connection with the No. G drilling machine manufactured by the Ingersoll-Rand Co. They may be built of wood or of steel pipe. For deeper drilling, however, the four-legged derrick is employed.

The tripod was observed, some five or six years ago, in extensive use in the Lima fields for the ordinary routine of pulling the tubing, and the cleaning of producing wells. It was made in heights of from 30 to 40 ft. In 1907 designs were made for a 40-ft. triangular derrick to be constructed of steel for use along the same lines.

Bansen in *Das Tiefbohrwesen*, Berlin, 1912, p. 274, illustrates a triangular boring tower which is in ordinary and regular use by the General Exploration Co. in connection with its large diamond-drilling machine. This tower, built of steel pipe, five panels in height, is double tapered, the apex panel being drawn in very quickly to a point under which the single large-diameter sheave is suspended. This type of drilling tower is said to be used in sinking wells to 1,200 m. in depth.

II. BRACED MAST

If the tripod is the simplest device for raising heavy weights in more or less permanent situations, the braced mast is the most convenient in

connection with portable or semi-portable drilling equipment. The uses of the braced mast go back to the very beginning of the art. It is impracticable to trace their history in detail; and only the main line of present-day equipment will be considered here. This has two branches: first, the mast in single stick, or built up and supported on the drilling frame, the whole structure being mounted on wheels and intended for transport by its own or other power (portable drilling machine); and second, the braced mast, usually built up in parts and supported on a solid foundation, the mast being in many cases independent of drilling mechanism (semi-portable drilling machines).

1. *Portable Drilling Machines*

It was several years after the sinking of the Drake well before engines and boilers were designed especially for the superior strain of drilling and handling oil wells and thus suitable for standard-rig drilling. The Wood and Mann portable outfit, in which the small engine was mounted upon the boiler, was the popular type for the shallow territory adjacent to Oil Creek, Pa. One of these was the cause of the great fire in Cherry Run Valley above Rouseville in 1866 (*Romance of American Petroleum and Gas*, Vol. I, p. 125, 1911).

Among the earliest manufacturers in this line was the Pierce Well Excavator Co., with machines worked out about 1880 to 1890, as pictured by Tecklenburg, Vol. IV. These were arranged for transport, and also for operation, by horse-power. The mast, braced back to the machine by two braces, carried a drilling pulley at its extreme top, and a sand-line pulley part way up. Drilling was done by cable without walking beam. About 1890, the appearance is noted of that type of drilling machine usually associated with the name Star, and manufactured, according to Tecklenburg, by Gould and Austin, with the observation that the chief purpose of this very effective apparatus was to bore for water in the dry, treeless plains of Western North America. The first condition was, therefore, to replace the high timbered boring tower by an easily transportable and yet solid boring mast. These machines were suitable for depths of 450 m.

a. Side Walking Beam, Center Post.—The drilling machines made by the Star Drilling Machine Co., Akron, Ohio, embody today the same essential features of a single center mast braced to the side of the structure, and a walking beam supported at the side of the machine, the whole mounted on a wooden frame, which carries, in addition, the operating machinery, boiler and engine, and is transportable on its own wheels. These machines are made for horse transportation or are self-tractors. In the smaller machines, made without walking beam, drilling is done by spudding with a jerk line drawn over the crown pulley (see Fig. 10).



FIG. 10.—STAR DRILLING MACHINE. HEAVY SERVICE TRACTOR.



FIG. 11.—ST. LOUIS CENTER-BEAM DRILLER.

The same structural features appear also in the Keystone driller, manufactured by the Keystone Driller Co., Beaver Falls, Pa., though in the larger machines of this type the mast is not set on the frame but is placed on a footing piece on the ground, or on the floor framing of the well.

b. Side Post, Center Walking Beam.—The St. Louis Well Machine & Tool Co., St. Louis, Mo., established about 1880, makes a drilling machine in which, as contradistinguished from the Star, the walking beam is mounted on the center, while the mast is placed on the side of the machine. The masts are usually built of wood, and the machines are made under the Robbins patent (see Fig. 11).

c. Braced Mast, Center Walking Beam.—This type of construction is an extension of the preceding one, and differs therefrom in that the mast



FIG. 12.—CYCLONE DOUBLE-BEAM DRILLER.

is now of the full width of the machine and the walking beam is suspended on stirrups, carried by a cross beam or suspended from the mast legs. This type of suspension is employed in the National mounted drilling machine. The Cyclone driller, manufactured by the Cyclone Drill Co., Orrville, Ohio, is in quite extensive use in all kinds of shallow-well drilling, mineral exploration, etc. The deep-well drilling machine manufactured by this company has double standard beams. These beams are supported on a samson post carried on the machine frame and reciprocate within the legs of the braced mast. The temper screw is hung from a cross bar, fastened to the outer end of the beams (see Fig. 12).

d. Braced Mast, Side Walking Beam.—The American Well Works, Aurora, Ill., was established in 1868 by M. T. and M. C. Chapman, well-

known names in the invention of drilling machinery, particularly the rotary. Many styles of drilling machines are made by this company, most of them utilizing the jerk-line method of drilling rather than the walking beam. In the American double walking beam rock-drilling machine, however, the walking beam is made double, the mast placed within it, and reciprocating motion imparted by two pitmans, one on each side of the machine. This machine is made in two sizes and will drill 8-in. holes to 1,000 ft. The largest machines are made with steam-driven tractors (see Fig. 13).

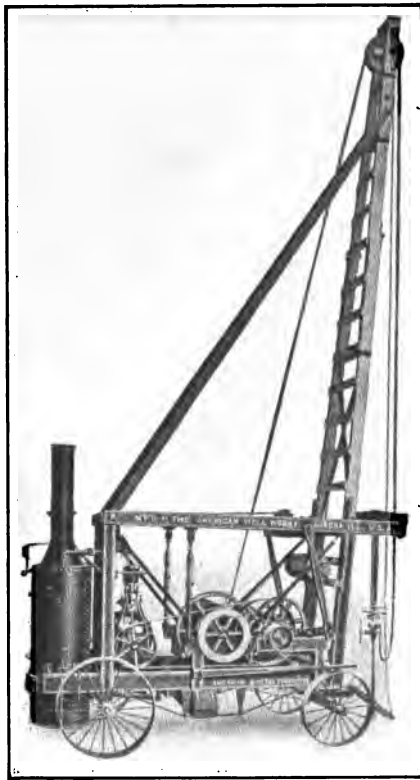


FIG. 13.—AMERICAN DOUBLE WALKING BEAM ROCK-DRILLING MACHINE.

The present Oil Well Supply Co. was organized in 1891 and is a continuation in lineal succession of the business established in 1867 by John Eaton, who is said to be the father of the oil-well supply business. The portable drilling machine manufactured by this company is the Columbia driller, formerly known as the Glenn driller, and is covered by patent No. 472,619, issued to H. S. and C. E. Glenn, Apr. 12, 1892, with later patents covering improvements. It is an all-metal machine. The mast is made of steel channels latticed together and to the full width of the

machine at the bottom, and the walking beams, built plate-girder fashion, are reciprocated by double pitmans outside of the mast. The No. 10 Columbia weighs 18,000 lb. without boiler, and has a normal capacity of 2,500 ft. The No. 1 or Columbia Junior drilling machine is intended for water wells, test bores or blast drills, and while made with a double-braced mast, is without walking beam (see Fig. 14).

e. The Modern Self-Tractor Drilling Machine.—The essential features of this type are shown in patent No. 295,413, issued Mar. 18, 1884, to James Gillott Martin, of Bradford, Pa. The object of his invention is to provide a drilling machine capable of conversion into a traction engine, by which the machinery may be moved in smooth or moderately hilly country. Indications in the description of this patent give approximately the date of the substitution of steam for horse-power in transportation.

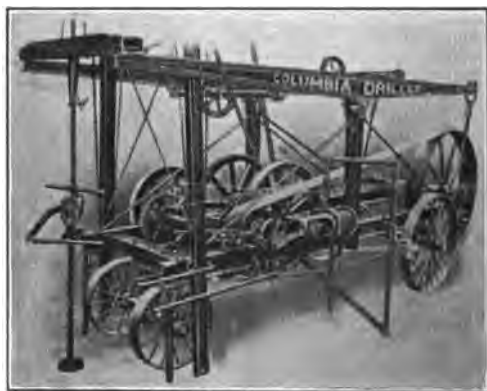


FIG. 14.—COLUMBIA DRILLER No. 10. LOWER PART OF MAST ONLY SHOWN.

2. Semi-Portable Drilling Machines

A semi-portable drilling machine was patented by I. Overall, Mar. 14, 1828, utilizing the braced mast with top crown pulley as the means of hoisting. Quite a number of the modern methods in operation are to be seen in patents Nos. 46,673 and 47,729, issued in 1865 to Walter Hyde, of New York, and attention should also be called to the improved rock drill patented by J. D. Dale, of Rochester, Feb. 27, 1866, No. 52,827; also patent No. 125,355, issued Apr. 2, 1892, to Fred S. Ward and Emmert Cooper, of Theresa, N. Y. This device is particularly significant from the fact that it employed a horse treadmill as motive power. Further improvements were made in patents Nos. 257,766 and 257,767, issued to Oscar Rust, of St. Joseph, Mo., May 9, 1882.

It is believed, however, that the semi-portable drilling rig of the present owes more to George Corbett, of Bradford, Pa., than to any other single person. Patent No. 385,241, issued to him June 26, 1888, discloses

a braced mast resting on a solid footing and carrying on its top a single sheave pulley for cleaning purposes and about half way up a larger pulley for spudding. It may be noted in passing that George Corbett is responsible for improvements in the standard drilling rig used with the four-legged braced tower, for instance as disclosed in patent No. 216,259, issued June 10, 1879. It is also to be noted that this same gentleman is responsible for the original design of the bobtail rig mentioned in discussing the special development of the steel drilling rig. The walking-beam rig ordinarily called a Corbett rig is disclosed in patent No. 396,544, issued Jan. 22, 1899, which shows in a general way a type which has come into very extensive use and consists of a wooden mast carrying two

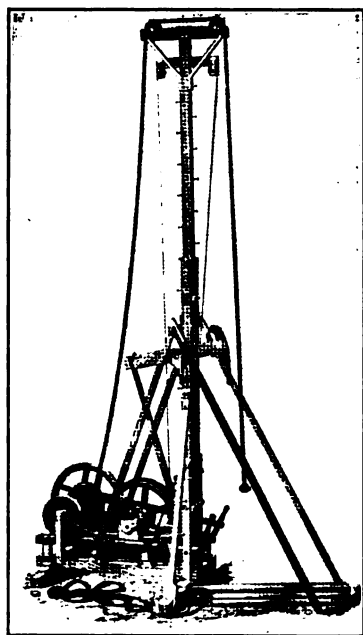


FIG. 15.—CORBETT SEMI-PORTABLE RIG.

balanced crown pulleys at its top, attachments for sand lines, and a double-braced stiff-legged pedestal. The band wheel, bull wheel, and sand reel are mounted on their own separate truck. The engine is separated from the other parts of the drilling mechanism. The band wheel is driven by a belt from the engine. The sand reel is operated by friction applied by lever; the bull wheel is also operated by friction or by a chain and clutch (see Fig. 15).

The Corbett rig was never built for wells much over 1,000 ft. in depth, but for shallow wells it came into extensive use, which continues down to the present time.

The National portable drilling rig, manufactured by the National

Supply Co., Toledo, Ohio, and Pittsburgh, Pa., is a development of the Corbett rig and is constructed under the improvements covered by Clyde Selwyn Wright, Quaker City, Ohio, in patent No. 737,545, dated June 13, 1903. The National Supply Co. was organized in 1894, succeeding Shaw, Kendall & Co., and the Buckeye Supply Co., which had been continuously engaged in this line of manufacture since previous to 1880. The No. 2 National driller can be depended on to drill wells 2,500 ft. in depth. It is built with a mast 60 ft. high, 39 in. wide at the base, in three sections. These masts, while furnished in wood, have also been made in structural steel. The National mounted drilling machines made by the same company embody the same essential structural features (see Fig. 16).

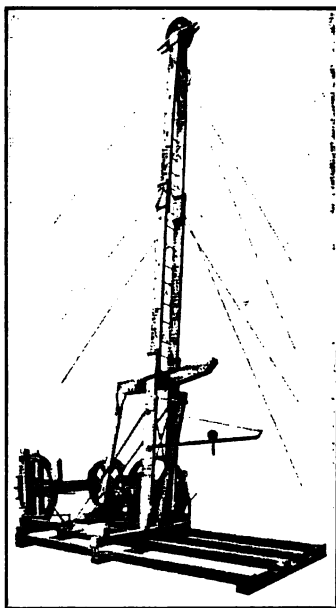


FIG. 16.—NATIONAL PORTABLE DRILLING RIG.

The same type of rig, known by the trade name of the St. Louis drilling rig, Rourke patent, is made by the St. Louis Well Machine & Tool Co., in accordance with patent No. 705,791, issued July 29, 1902, to James Rourke, of Parkersburg, W. Va.—with this difference, that in the National driller the walking beam is suspended between the legs of the mast, whereas in the St. Louis drilling rig it is mounted on its own independent samson post.

A machine of analogous character is that known as the Parkersburg chain-drilling machine, which is made with a mast 58 ft. high and is capable of drilling wells 2,000 ft. or more in depth. The operating mechanism, walking beam, and band wheel of this rig, mounted on the

same kind of frame, may also be used with a standard four-legged bolted derrick and when so used is known as the Parkersburg bull-rope drilling machine. The chain-drilling machine made by the Parkersburg Rig & Reel Co. was introduced about 1902.

A noteworthy drilling rig of the same general character (patent No. 668,209, Feb. 19, 1901, to Charles Denio Pierce, of Jersey City, N. J.) need not detain us further here except to note that the braced mast was to be constructed of structural steel and the walking beam was to be of the same material.

3. *The Knupp Rig*

This rig, patented Mar. 14, 1905 (No. 784,571) by Jacob C. Knupp and James G. Green, of Warren, Pa., was intended to possess portability as well



FIG. 17.—THE KNUPP SEMI-PORTABLE RIG.

as stiffness. The base of the braced mast had been widened. It was also stiffened triangularly by braces and guyed with the usual guy lines. The band wheel, sand reel, and walking beam were placed on their own independent foundation as in the Corbett rig, and this type of construction has been used in drilling wells as deep as 3,500 ft. There were unfortunate circumstances in connection with the operation of several of the rigs. Sufficient care was not taken to guy the derricks properly and

some of them were pulled over. It is possible that a dozen would cover the number of these rigs actually placed in use about 1905. The writer is somewhat identified with the Knupp rig for the reason that in 1903 and 1904 an endeavor was made to work up a steel structure to incorporate the lines of the Knupp construction. These propositions, however, came to naught (see Fig. 17).

III. THE FOUR-LEGGED BRACED WOODEN DERRICK

I am unable to say when the four-legged braced wooden derrick first came into use as a hoisting tower for well sinking. It is probably an evolution of the type of head frame customary in sinking mining shafts generally; and its form is so well established in the history of the art as to

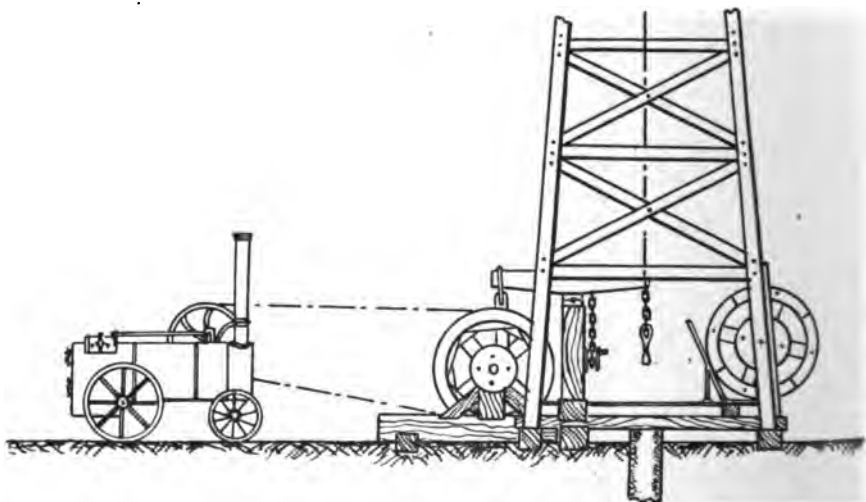


FIG. 18.—CANADIAN DRILLING RIG, 1866. (W. H. MACGARVEY.)

need no extended comment. It appears, for example, in a patent without number issued Mar. 24, 1825, to L. Disbrow, and in a more elaborate form in a patent to the same party issued Nov. 1, 1830. Unfortunately the descriptions of these two patents are not in my hands; but I note that the legs were made of square timbers, and that the girts were mortised and inserted into the legs by the use of keys, so as to provide for dismantling; also, that in 1825 the drilling line was run up over the crown pulley and attached to a treadle so as to permit drilling in treadmill fashion; and that hoisting was done by a two-handled windlass with a fly-wheel. The four-legged braced derrick made with squared timbers also appears in patent No. 47,868, issued May 23, 1865, to John Y. Smith, of Alexandria, Va.; in No. 48,006, issued May 30, 1865, to D. H. Wiswell and George W. Shaw, of Buffalo, N. Y.; and in No. 52,642, issued Feb. 13, 1866, to

S. J. Goucher, of Philadelphia, all of which indicates that by that date the framed derrick was in such extensive use as to need no further description in patent specifications (see Fig. 18).

In the *History of Pithole*, written by "Crocus" (Charles C. Leonard) and published in 1867, it is said in description of well-sinking methods:

"First comes the derrick, a lighthouse-looking structure, some 48 ft. in height, and a temporary shed to cover it and shelter the operators and afford them sleeping quarters as it is not safe to leave the premises or machinery for even a night. If they did, they might find every movable thing had disappeared. Even the derrick and engine have been stolen in a single night and not long since an enterprising thief stole the tubing out of the bottom of the well as fast as the owner let it down to the bottom. In this manner he purloined nearly a thousand feet of tubing from a six hundred foot well."

It is true that the early oil-well derricks were constructed of poles cut from the adjoining forest and framed together in heights of 28 and 30 ft. by the use of spikes, wooden pins, and other crude devices. The ladder going up one side of the derrick was constructed by means of cleats nailed to one of the derrick legs. The essential features of girts and bracing are, however, shown on photographs taken at Pithole, Tidioute, Cherry Run, and other early oil fields in western Pennsylvania. Modifications in the four-legged derrick have been due essentially to their increase in height and to the use of sawed timber put together by carpenters instead of rough timbers cut out of wood and framed by inexperienced hands. That there is so little modification in the structure since the date of the Drake well is, beyond question, due to the fact that the essential features of the structure had already been worked out in the explorations of the salt industry.

It is, however, important to history that the four-legged braced derrick appeared on the labels pasted by Samuel Kier, of Pittsburgh, on the bottles of petroleum which he sold in 1848 and later for their wonderful medicinal virtues and that it was the sight of one of these labels and the words 400 ft. thereon which gave George H. Bissell the inspiration to undertake the organization of the Pennsylvania Rock Oil Co., under which organization Edwin L. Drake drilled the famous well at Titusville. McLaurin's statement in *Sketches in Crude Oil*, p. 81, is:

"The first derricks were of poles, twelve feet base and twenty-eight to thirty feet high. The ladder was made by putting pins through a corner of a leg of the derrick. The Samson-post was mortised in the ground. The band-wheel was hung in a frame like a grindstone. A single bull-wheel, made out of about a thousand feet of lumber, placed on the side of the derrick next to the hand-wheel, with a rope or old rubber-belt for a brake, was used. When the tools were let down the former would burn and smoke, the latter would smell like ancient codfish."

IV. THE BOLTED WOODEN DERRICK

The advantage of wood in the construction of derricks and drilling rigs is that it is very cheap and can be made to meet any condition by building up pieces. If for any reason the rig builder makes a derrick too light and it begins to fall in, or rack to pieces, any one can take a piece of plank and nail it on in the right place to save the day. If the amount of wood used is too great for the load it has to sustain, it does not make any particular difference and does not add a great deal to the cost of the outfit, the large item of expense being in the carpenter work rather than in the material itself, though today it behooves us to pay a great deal more attention to the economical use even of wood.

This essential consideration of economy is not an absolutely new thing in this year of our Lord 1915. Beyond question, it lies back of the substitution of sawed square timber for poles and in the subsequent substitution of 2-in. plank in the legs for solid sticks of timber. We are prone to believe that the present generation invented the conservation of materials. The introduction of plank in derrick construction, say about 1865, is at once a blow to our pride and an indication of a correct though possibly unwritten appreciation of the magnitude and character of drilling stresses.

It has already been noted that L. Disbrow in 1830 intended to have his derrick put together with mortises and keys. Samuel S. Fertig, of Titusville (patent No. 130,706, Aug. 20, 1872), describes a construction and arrangement of the parts of a sectional derrick frame, whereby said frame after being built may be readily taken apart, separated into sections, or deprived of one or more sections by the removal of a few bolts without injury to the timbers, and in such manner that it may be speedily rebuilt. Fertig's derrick was made with staggered joints in two-panel lengths, with the corners, braces, and girts of plank, each part-panel piece being put together in sections and removable by bolts at the field. The use of bolts is specifically referred to in well-drilling apparatus patented by William Lowman, of Marionville, Pa., No. 388,889, Sept. 4, 1888, and is also disclosed in the patents of George Corbett. By 1892 the bolted derrick had reached such a state of development as to be actively exploited by the Oil Well Supply Co. and shown as a specific article of manufacture on p. 12 of their catalogue with the comment that it is specially adapted for being moved from place to place; the various pieces being numbered and the parts which join being specially marked. In other words, there had been worked out by that date a system of standardization which would permit accurate manufacture and accurate erection. The purpose of the invention is said to be to construct, easily assemble and disassemble a boring tower, which may be sent in separate parts to timberless regions, where it can be reassembled with facility.

The exploitation of gas has provided a field for their use much more extended than contemplated by the original invention, and the bolted derrick is, therefore, one of the most modern tools of the drilling contractor. Those made by the Parkersburg Rig & Reel Co. are perhaps in more extensive use today than any other (see Fig. 19).

The most recent development in this line is what is known in Oklahoma as the turnbuckle derrick, which is a derrick built with wooden corner posts and girts, and with iron rods fitted with turnbuckles for



FIG. 19.—PARKERSBURG BOLTED DERRICK. TYPICAL PENNSYLVANIA CONSTRUCTION.

diagonal braces. There was said to be a growing demand for such derricks in 1913 and they are, beyond question, suitable for use in that shallow territory and for pumping purposes, in spite of the fact that the rig builders and drillers have not, as a rule, taken kindly to the use of turnbuckles. One of the most recent patents on construction of this character was No. 798,021, issued Aug. 22, 1905, to Henry Eck, of Nobleville, Ind., which seems to cover the essential features, except that the legs are made of square timbers instead of planks.

V. PIPE DERRICKS

It is natural that the attention of operators should be turned early to the use of steel pipe since there are to be found in all oil and gas fields great quantities of 2, 3, and 4 in. pipe which has survived its usefulness in carrying oil, gas, or water under pressure, but is still good for structural purposes, round sections being well fitted theoretically to sustain compressive stresses. The difficulties in the construction of derricks did not

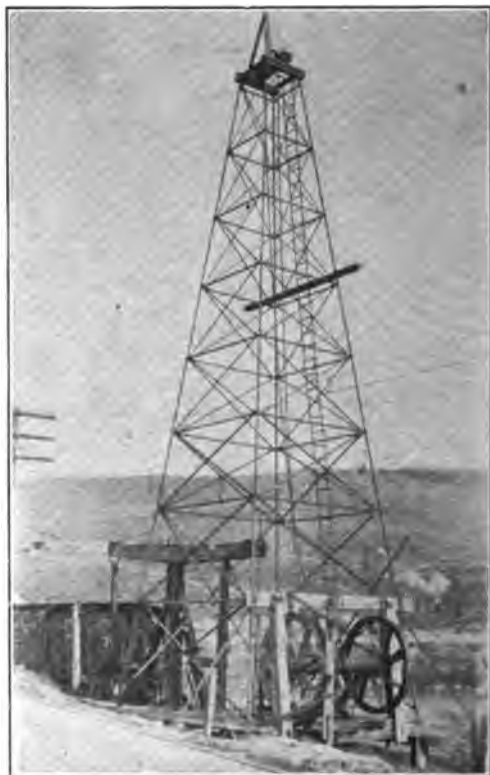


FIG. 20.—NEILL PATENT PIPE DERRICK. MACDONALD, PA., 1909.

lie in the legs but in the other parts. Pipe is not fitted for crown blocks, or ladders, or even for the heavy girts which take the tops of the bull-wheel and calf-wheel posts. In the most modern types of pipe derricks, these parts are made of steel beams, angles, and bars. Nor is pipe suitable for bases, though it may be said, as a matter of record, that Zahniser Brothers & Sten Co., of Washington, Pa., endeavored in 1910 to make this use of it, and constructed one or two bases.

Bent pipe has also been considered for walking beams, as shown in patent No. 914,608, issued to Hezekiah Shannon, of Cyclone, an illus-

tration of which will be given under the general heading of "Structural Steel Drilling Rigs."

Pipe is suitable for band, calf, and bull wheel shafts, but with these exceptions is, as a general rule, inadequate for the construction of more than the derrick proper. True economy requires the use of each material in the service for which it is adapted; and the use of pipe for derricks is recommended by structural fitness. The same consideration bars its use elsewhere in this line, just as it has been long since definitely abandoned in structural fabricating shops, because experience shows structural steel shapes to be more economical than pipes when both have to be made of new materials. Fig. 20 shows the Neill pipe derrick.

1. *Early Endeavors to Introduce Pipe Derricks*

Many improvements in the mechanism of well drilling were made in the early days by Jesse Button, of New York City, whose patent No. 210,007, dated Nov. 19, 1878, describes a wooden frame, constructed so as to support compactly the different parts of the machinery used in drilling wells and carrying a removable tower for lifting and lowering the drill rods. This removable tower might be made of wood or metal, but Mr. Button preferred to construct it of metal tubing, the vertical supports and cross braces being made with screw joints, so that it might be taken apart and used or not, according to the local conditions. The lower bottom panel, made of wood, was built with vertical posts, probably for the same reason as influenced the writer in the first design of the short-panel steel derrick, mentioned hereafter. The clamps at the joints are not fully described, but would probably be made of cast iron.

Patent No. 331,714, Dec. 1, 1885, was issued to Albert T. Hyde, of Oil City, Pa., for improvements in artesian-well rigs, with special reference to oil-well drilling. Mr. Hyde proposed a triangular derrick, the three legs of which were made of tubular posts, and in which the horizontal members or girts were made of pipe and the diagonal members of rods, the ends of both girts and rods being forged out flat and drilled for attachment to the legs through flanges and couplings, as shown in Fig. 21. Mr. Hyde's idea was to use one leg of the derrick to replace the samson post and so to compress the parts of a drilling rig as to place them all on one foundation about the size of an ordinary derrick floor with necessary modifications in the arrangement of engine, band wheel, bull wheel, etc. The walking beam of this rig was to be in two parts, one fastened to the derrick leg and the other to the derrick floor, and operating like a modern pumping jack.

The pipe derrick does not appear in the Oil Well Supply Co.'s catalogue of 1892, which may be taken as fairly covering the state of the art at that date. It does appear, however, in the 1900 catalogue, constructed

in accordance with patent No. 346,466, granted to George Corbett, of Bradford, Pa., Aug. 3, 1886. Joints of this derrick were made in slip sockets and keyed on—no threads being used except on the turnbuckles of the stay rods, which were made of iron or steel rounds. At the joints, holes were cored in the castings to receive the ends of the diagonal rods on which dependence was necessarily placed for holding the structure together. Special castings were also provided at top and bottom for the attachment of crown block, etc.

A construction, in some respects more modern, is disclosed in patent No. 566,364, issued Aug. 25, 1896, to Amos C. Wilson, of Butler. The coupling sleeves are provided with flanged T-slots and the girts are provided with screwed-on T-heads adapted to fit the T-slots of the coupling

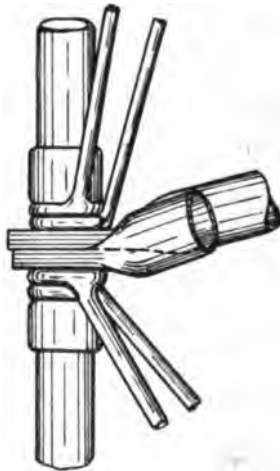


FIG. 21.—HYDE PIPE-DERRICK JOINT.

sleeves, somewhat like the way in which the cheaper grades of iron beds are constructed. Projections are formed within the circular portion of the couplings, against which the leg sections abut. Diagonal cross braces are screwed into socket braces which are connected to the coupling sleeves, through ears in their upper and lower surfaces, by bolts.

2. *More Recent Pipe Derricks*

It is impracticable at present to ascertain exactly what number of these derricks were actually constructed. It is likely, however, that some few of them came into actual use in spite of their crude design and expensive construction, just as it is likely that a few wind-mill towers were made under patent No. 490,267, issued Jan. 24, 1893, to William H. Burnham and John H. Miller, Batavia, Ill., assigners to United States Wind Engine & Pump Co., of the same place. These wind-mill towers were

triangular and put together with 60° structural angles, clamps, and rod braces.

The use of U-bolts with or without forged clamps for fastening members of the derrick together appears also in derricks that have actually been erected and numbers of which have come into use in spite of their crude form and their relatively high cost.

At Supulpa, Okla., in October, 1909, an attempt was made by a Mr. Barnes to introduce pipe derricks, one of which, of very crude design, with screw joints and badly forged end connections, was on exhibition at Kiefer. About the same time the manufacture of pipe derricks was carried on at Washington, Pa., by Zahniser Brothers & Sten Co., who declared in their advertising literature that they had limited the use of bolts to a minimum in the construction of their make of derrick, that there were only 16 bolts used in the crown and 16 in the base, and the derrick proper could be made complete without the use of bolts; but it was believed that the use of bolts in the crown or water table made the derrick easier to take apart, if it was to be moved, as is often the case in a dry hole or gas well; and that this derrick could be taken down and rebuilt in half the time required by any other pipe or steel derrick on the market.

It was also claimed that the derrick could easily be put in perfect alignment by the use of adjustable rods (made with turnbuckles), and that it had no weak spots, since the pipes were not flattened at the ends, as in other derricks. This company also manufactured a trussed walking beam and a derrick base made of steel pipe. The design of the derrick corner coupling is similar to that patented by Amos C. Wilson but in simplified and more practical shape. Legs and girts butt against divisions in the hollow portion of the casting, and the whole structure is held together by the diagonal braces connected to bolt holes cored in the up-standing ears of the corner casting itself, as shown in Fig. 22.

Derricks made largely but not exclusively of pipe were constructed on a small scale by Simpson Brothers at Oakdale, Pa. The legs were connected by ordinary screw couplings. Girts were made of pipe flattened down 9 in. or more at each end, and with the extreme end of the flat portion bent in a quarter circle to engage the corner leg, to which it was bolted by a bolt passing through two girts, leg, and coupling. Diagonal braces were in like manner made of pipe flattened down at the ends, and connected not to the clamp but to a girt by single bolt passing through two braces and the girt (see Fig. 23). The drawing in my possession shows a 59-ft. pumping derrick only. It is obvious that this method of construction is not adequate for derricks made for drilling purposes. The ladders for these derricks are made of wood, as also are the crown blocks and bases.

Many of the derricks in use in the Evans City town-lot development, it is understood, are being made by Jacob Morris at Mars, Pa., but I cannot now say exactly what type of construction Mr. Morris follows.

Pipe derricks are also made in the Pittsburgh district by Frank Kinney, boiler maker, at Oakdale, Pa.; and the activity of these smaller shops and shops owned not by manufacturers but by oil companies themselves, is, no doubt reflected in other places throughout the oil and gas fields, where the crude methods of manufacture are counterbalanced by the low cost of second-hand material.

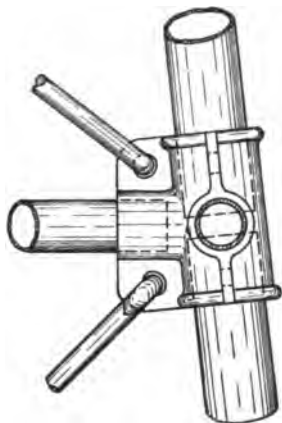


FIG. 22.—ZAHNISER AND STEN PIPE-DERRICK JOINT.

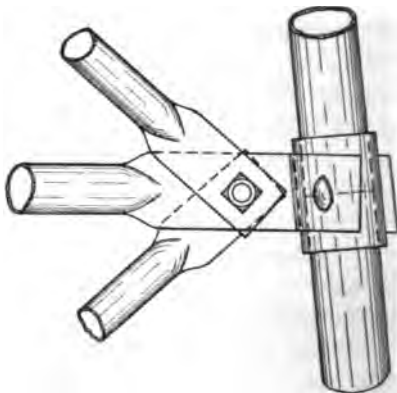


FIG. 23.—SIMPSON PIPE-DERRICK JOINT.

Among the noteworthy endeavors to utilize pipe in the construction of steel drilling rigs and derricks is a derrick made for his own operations by Godfrey L. Cabot, of Boston, Mass., the well-known manufacturer of carbon black. The method of joint construction used by Mr. Cabot is shown in Fig. 24, in which it will be noted that the splice is a malleable

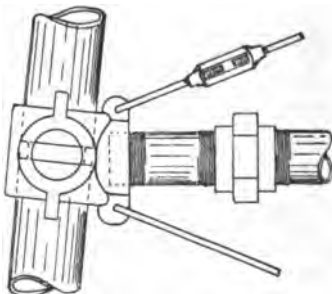


FIG. 24.—CABOT PIPE-DERRICK JOINT.

casting, with inside rings, against which the 4-in. leg sections bear, and with outside wings to which the $\frac{5}{8}$ -in. diagonal rods are attached, hook fashion. The girts are made of 2 and 3 in. pipe and are connected to shoulder nipples screwed into the joint castings by threaded ends and unions. Turnbuckles are used on the diagonal braces; and these, in

connection with the unions, permit a great degree of adjustability in the members, which obviates the necessity for very accurate cutting to length and at the same time permits the maintenance of alignment. One of these pipe derricks erected by Mr. Cabot on the McGeary farm, some 2 miles south of Worthington, Armstrong County, Pa., has been used in drilling five holes.

3. Neill Patent Pipe Derrick

The development of the pipe derrick as a large commercial enterprise is due to the successful use of the forged steel clamp patented in the first instance by T. A. Neill, Field Superintendent for the South Penn Oil Co. In 1908 it occurred to Mr. Neill that a steel derrick suitable for pumping and cleaning wells could be successfully fabricated of second-hand pipe.

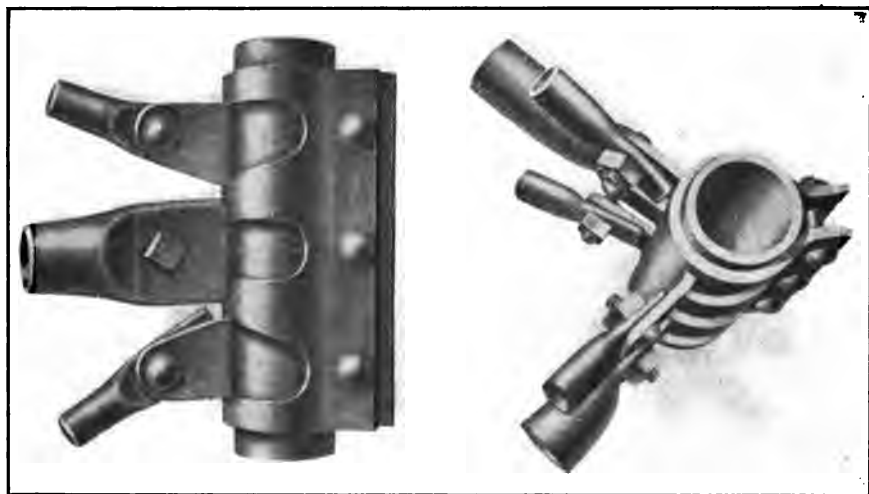


FIG. 25.—NEILL PIPE-DERRICK CLAMP.

He constructed a small wooden model indicating his idea of a clamp for the purposes of joining two pieces of pipe together without a thread and at the same time reinforcing the joint. This model was turned over to Lee C. Moore of Lee C. Moore & Co., Inc., and the first 80-ft. derrick was built in Mannington, W. Va., with clamps made of cast steel for joining the leg sections together. This first derrick demonstrated not only that the derricks would prove a success for cleaning purposes but also for heavy work as well, and Mr. Neill proceeded to cover his invention by letters patent No. 933,386, dated Sept. 7, 1909. Rights of exclusive manufacture passed to Lee C. Moore & Co., with reservation of a shop license to the South Penn Oil Co. for the construction of derricks for its own uses. Fig. 25 shows the clamping device.

During 1909 there were made and used by the South Penn Oil Co.

and the manufacturers more than 20,000 cast-steel clamps, which were sufficient for approximately 500 derricks. Their use demonstrated, however, that cast steel was not uniform in strength and it was concluded to adopt rolled steel for the purpose. Accordingly in 1909 tools were designed to make the clamps by the drop-forging process, and since that date they have been so made. Improvements in their manufacture are covered by patents No. 1,015,821 and 1,082,207, issued to Lee C. Moore Jan. 30, 1912, and Dec. 23, 1913. The substitution of wrought steel for cast steel reduced the weight of the clamp and its cost without any decrease in the strength of the derrick at the joint, and thus largely guaranteed its success.

While the manufacturers have distributed a great many derricks made of new pipe, the more usual practice is to sell the clamps to users for the fabrication of their own derricks; and during the year 1913 the manufacturers produced sufficient clamps to construct 1,500 derricks, and that in spite of the fact that the lumber yards back of the rig builders have opposed the introduction of steel, either pipe or structural, in the construction of drilling rigs.

The extension of the Neill patent pipe derrick into rotary-drilling territory, and its use in greater heights than are required in the Appalachian oil and gas fields, made necessary the reinforcement of the pipe itself. Tests were, therefore, made under the auspices of the Bureau of Standards at Pittsburgh, which demonstrated the safe carrying capacities, under compression, of derrick leg sections built by inserting one or more tubes inside the outer section. This made it possible to increase the strength of the derrick without the use of large-sized pipe not easily obtainable and also without the multiplication of a great number of clamps, each size of which requires its own special forging equipment. It is possible to build up derricks in this manner of tubular leg sections to 30 lb. per ft. with a steel area of 8.3 sq. in.

Clamps are now made to fit all sizes of pipe from 2 to 6 in. inclusive. The heaviest derricks made of tubular sections have been of the triplex leg-section pattern (three pipes, one within the other) weighing 28 lb. per foot, 117 ft. high, 26 ft. base and 5 ft. 6 in. top. The most popular styles of this type of derrick are the 59-ft. pumper and the 80-ft. driller. These are built regularly with pipe legs, girts, and braces, tops and crown blocks of rolled-steel beams, ladders of flat steel bars and round rungs, with steel I-beam bull-wheel girts and steel channel bases, the designs for crown blocks and bases being the same as on the Woodworth derricks. By arrangement between respective manufacturers, the pipe derrick, as will be noted later, is used almost interchangeably with the structural steel derrick in the construction of complete steel drilling rigs.

The essential features of the Neill patent pipe derrick are: The design is flexible because the strength can be increased to cover any requirement

by the use of heavier pipe for the leg section or by the duplex or triplex method of construction, and this without change in clamps. There are no screwed joints, and the derrick is assembled with clamps and bolts. The leg sections are plain, while girts and diagonal braces, which are also of pipe, are swedged down at their ends to provide a flat bearing surface against the clamp and proper width of material for punching the holes through which they are connected to the clamps themselves. U-bolts connect the girts and diagonal braces at their intersection.



FIG. 26.—STEEL DRILLING RIG, PIPE DERRICK, 1909. SOUTH PENN OIL Co., SHINNSTON, W. VA.

It is obvious that in this type of construction the overturning of the derrick under wind or its pulling in by eccentric distribution of loads is resisted entirely by the friction of the clamp on the pipe and in consequence the bolts should be well tightened. Nine bolts are used at each joint in the 80-ft. driller and the spring action of the clamp has been found by experience to develop all the strength required. The pipe derrick is, therefore, no longer an experiment, more or less crude, but a fully developed engineering structure. Fig. 26 shows an 80-ft. derrick of this kind.

VI. STRUCTURAL STEEL DERRICKS WITH LONG PANELS

The first derricks made of structural steel shapes were designed by men more familiar with bridge and building construction than with the practice of deep drilling. Inasmuch as the structures were supposed to remain permanently in position and to be used as structures rather than tools, convenience in erection was assumed to be a matter of secondary importance. This failure of the structural steel designer to appreciate

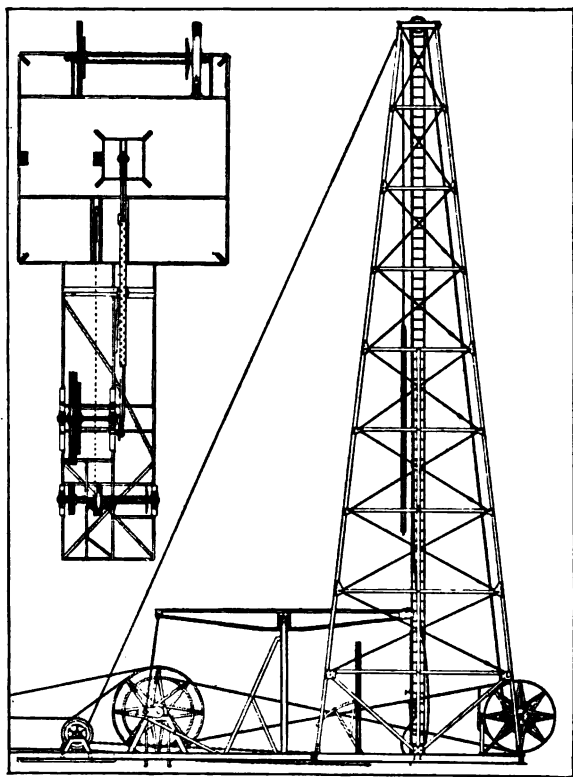


FIG. 27.—STRUCTURAL STEEL DRILLING RIG. O. W. S. Co. CATALOGUE, 1892.

the needs and desires of the driller has had somewhat to do with the slowness of the substitution of steel for wood.

The first appearance of the structural steel derrick known to the writer is on pp. 14 and 15 of the illustrated catalogue issued by the Oil Well Supply Co., in 1892, which shows steel derricks 72 ft. high, made with steel angle legs, steel base and crown block, the diagonal members being made of rods adjustable by the use of turnbuckles. Page 14 shows the ground plan and side view of a complete steel rig with steel walking

beam, samson post, foundation, and sectional wheels. The description of the derrick is:

"It is practically indestructible, made of light pieces easily transported and handled, which are fastened together with bolts; every part is made of steel; power is applied by a rope running through double grooves in engine pulley, band wheel and sand reel head. The first rig of this kind was sent to Australia for use in a district where wood decayed rapidly."

The transmission from engine to band wheel was by rope drive, and the sand reel was not driven, as usual, by a friction pulley in contact with the rim of the band wheel, but by a rope from the engine. The band wheel was, therefore, fitted with grooves instead of a plane surface. The impression made by the illustration is that the entire structure was primitive in form. The panels, however, were made in short length, nine in number, for the 72-ft. derrick (see Fig. 27).

The derrick reappears in the Oil Well Supply Co.'s catalogue, edition of 1900, p. 13, and on p. 450 of a catalogue issued in 1897 by the National Tube Works Co. in French and English, and intended for circulation abroad, with the additional information that the pieces were fitted, numbered, and painted; that the parts presented but little surface to the wind; that the wind strains had been carefully considered and that there was no danger of the wind blowing the derrick over; that the derricks were made of any height from 40 to 72 ft.; that the construction was particularly adapted for use in hot countries; and that the "first steel rig (all the parts being of metal) was sent to Australia to be used in a district where wood decayed rapidly. The outfit gave complete satisfaction and drilled a number of wells." The wheels for this type rig were exhibited at the World's Fair, Chicago, in 1893. In 1894 a second rig was built and shipped to Teheran, Persia, via Aden. Hauling from port to Teheran was via camel train.

The year 1903 marks a new endeavor to introduce the structural steel derrick. In that year the Pittsburgh plant of the American Bridge Co. built derricks in heights of 50, 60, and 70 ft., tower fashion, with three, four, and five panels respectively; that is to say, with panel heights, on an average, of 17, 15, and 14 ft. The weights of the derricks were approximately 9,000, 11,000, and 12,500 lb. The legs, braces, and girts were all angles. The ladders were made of wood. These derricks had buttresses at the base on all four sides to prevent overturning and needed for their erection gin poles or other similar scaffolding. They looked well on paper and the designer, no doubt, considered them an excellent substitute for wood. Many of them were built and some are now to be found over pumping wells in the McDonald, Pa., territory (see Fig. 28).

The same year the Chester B. Albree Iron Works, of Pittsburgh, Pa., built two or three structural steel derricks on the order of the National Supply Co. for shipment to Vera Cruz, Mexico. These derricks were 70

ft. 8 in. high under the crown block, with a base of 21 ft. 8 in., and with buttresses of the same general character as those made by the American Bridge Co. They also had an average panel height of about 14 ft., but the ladders were of steel flats for sides and round rods for rungs (see Fig. 29).

In the same year the Carnegie Steel Co. built for the South Penn Oil Co. and the Carnegie Natural Gas Co. 12 derricks, which were used in the Pennsylvania and West Virginia oil fields. They were 80 ft. in



FIG. 28.—LONG-PANEL STEEL DERRICK, 1903. BUILT BY AMERICAN BRIDGE CO. ERECTED AT McDONALD, PA.

height with a 20-ft. base and were likewise constructed along the lines of the structural steel tower, the 80-ft. derrick being built in seven panels averaging a little less than 12 ft. in height. The derrick and base weighed 24,000 lb., and provision was made at the top for a two-pulley crown block. They proved to be stiff and steady under strain; and in spite of their long panels were removed from their original location and re-erected in the drilling of more than one well each. They were probably strong enough to drill a well in this territory of a mile and a quarter deep, or

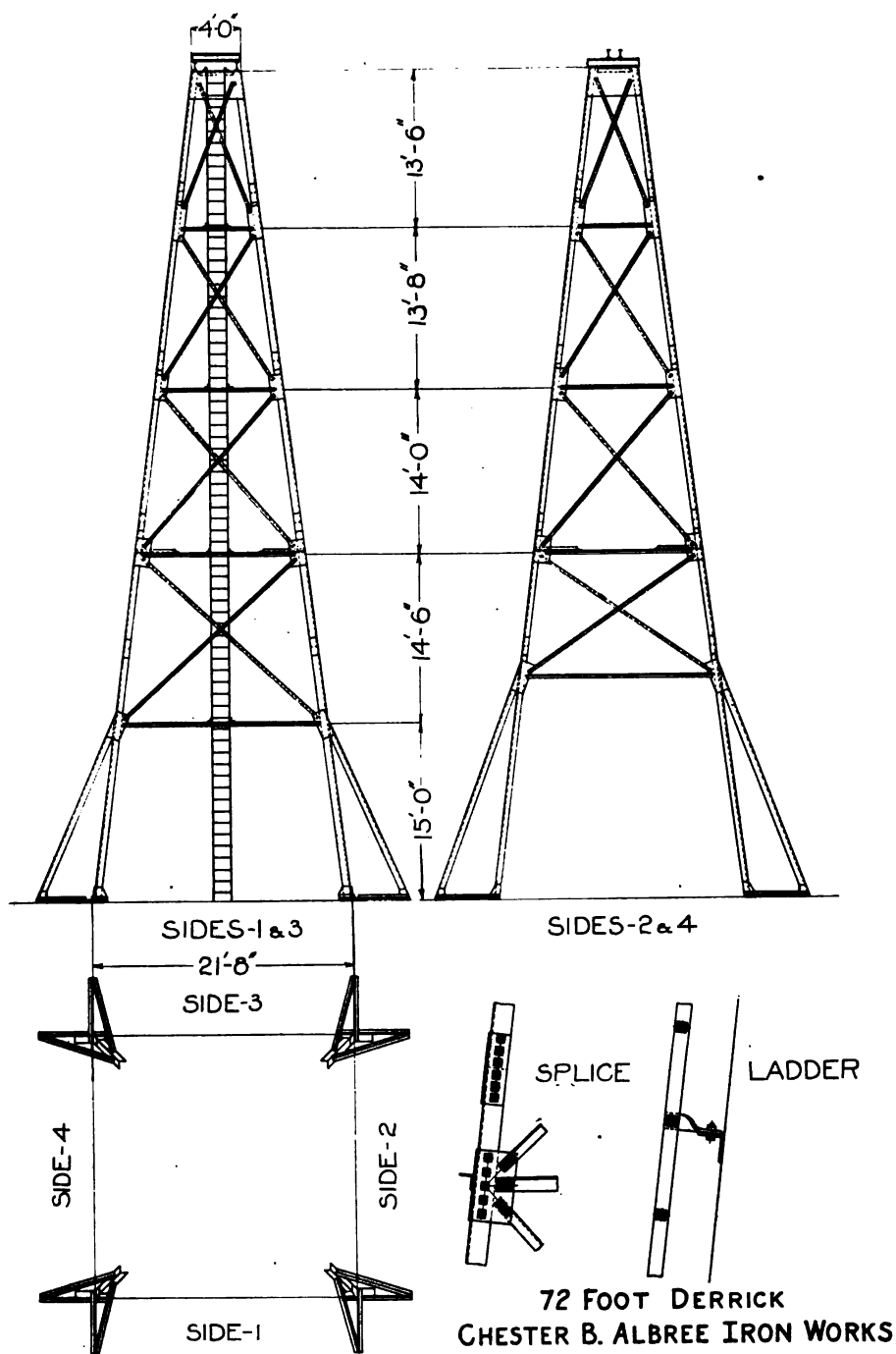


FIG. 29.—LONG-PANEL STEEL DERRICK, 1903. DESIGNED BY CHESTER B. ALBREE IRON WORKS.

about twice as deep as any well over which they were ever erected. It may be noted in passing that the 80-ft. structural steel derrick as today constructed is built with ten panels and weighs, including crown block, ladder, and base, in the neighborhood of 15,000 lb., and will drill as deep wells as any of these 12 ever undertook (see Fig. 30).

No endeavor was made in the construction of these derricks to design a complete drilling rig. The next step in this direction was taken in 1904



FIG. 30.—LONG-PANEL STEEL DERRICK, 1903. BUILT BY CARNEGIE STEEL CO.
ERECTED BY SOUTH PENN OIL CO., SHINNSTON, W. VA.

when, in conjunction with the Oil Well Supply Co., the Carnegie Steel Co. designed a standard steel oil derrick, California type, 72 ft. high, with a 20-ft. base, the intention being to furnish, in addition to the derrick proper, the necessary machinery supports for bull wheel, calf wheel, band wheel and sand reel, and so to detail each part as to permit the use of rig irons, wheels, etc., on the steel structure without substantial divergences from parts carried in stock by oil-well supply stores and designed primarily for use on wooden structures. This derrick was likewise

made tower fashion with six panels of an average height of 12 ft (see Figs. 31 and 42).

The first two derricks constructed after this design were shipped in January, 1908, one to Peru and the other to Argentine, after the lower panel of one of them had been erected complete in the shop for the proper adjustment of all working parts. In addition to the derrick, the order included base, machinery supports, walking beam, and house framing as far back as the sand reel, and each outfit complete weighed 34,000 lb.



FIG. 31.—SOUTH AMERICAN STEEL DRILLING RIG, 1908. SHOP ASSEMBLEMENT OF BOTTOM PANEL.

They were shipped to New York by rail, thence by steamer to South America, and were hauled inland from 300 to 400 miles (see Fig. 32).

The performance of the Argentine rig is clearly shown in the report of the driller, written from Zapala Sept. 6, 1908, and illustrates the vicissitudes which may be encountered and successfully overcome by a well-built steel structure:

"I got the rig set up over the Acme Oil Company's drilling, a little over 1,100 feet deep, and drilled to a little less than 1,500 feet deep, when we had an English expert come out here to look over the ground and he decided on another location, and then we had to move the rig. This we did without taking down the derrick. We disconnected the sills under the sampson and jack posts from the derrick and moved each of them

intact. The derrick we moved on 4" pipe rolling on 3 x 6" plank and pulled with block and tackle, using two yoke of oxen for motive power. The band wheel was slung on two two-wheel carts and pulled by eight oxen. The boiler was also mounted and hauled by four yoke of oxen. We moved the whole rig in four and a half days and in two days more had begun work. The derrick can be tumbled around any way and does not spring or rack in the least. It surpasses all of our expectations."

The third derrick of this design was sold to the Oil Well Supply Co. for shipment to the Burmah Oil Co., Ltd., at Rangoon.



FIG. 32.—72-FT. LONG-PANEL STEEL DERRICK. BUILT BY CARNEGIE STEEL CO., 1907. ERECTED AT ZAPALA, ARGENTINE, 1908.

VII. SHORT-PANEL STRUCTURAL STEEL DERRICKS

Experience with these derricks indicated, first, that the structures were much heavier than the operating conditions in the Appalachian oil and gas fields required; and second, that the re-erection and subsequent re-use of the derricks would be very much facilitated if the panels were made short, approximately as in the wooden rig. It was further observed that, with reduced panel heights, the same strength in the legs might be obtained by the use of thinner and lighter sizes of angles (the strength of such sections

in compression being determined by their ratios of slenderness), and the bracing made more effective.

Derricks were, therefore, designed in 1908 with short panels; and the modifications which have been made in them since have been made with a view, first, to suit the structure as exactly as may be to the wishes, and even the prejudices, of the driller, and second, to give them such strength and stiffness as was required in the light of careful observations on their use in different fields.

The designer had learned that, after all, this field of manufacture is a very conservative one; that commercial success lies in giving the driller an exact substitute for the wooden rig to which he has been accustomed; and that it is impossible to do anything more in the way of standardization than to follow only the broadest lines. Since that date the Carnegie Steel Co. has manufactured derricks to the established patterns of standard, California, and rotary types and has shipped them, on orders placed direct with the company or with dealers in oil-well supplies, to practically all the well-known fields with the exception of Europe and the Pacific Coast of the United States.

In the working out of these principles the designer has not hesitated to deviate from engineering practice as embodied in other lines of structures. He has not failed to abandon what he thought good for that which the drillers considered better. Derricks have been made as low as 40 ft. and as high as 106, with safe drilling loads from 40,000 to 300,000 lb., with capacities for the very shallowest pumping and the very heaviest drilling. Their essential feature has been only the use of structural steel wherever possible in their construction.

The short-length structural steel derrick has been manufactured with two kinds of bottom panels, in a variety of types and sizes, and with different kinds of joints. Under these three general divisions the steel derrick will be treated.

1. Bottom Panels

The three 72-ft. long-panel steel derricks built in 1908 deviated from the wooden derrick in that the bottom panel was made vertical instead of tapering continuously from crown block to base. The earlier designs as noted had buttresses at the bottom, because designers considered it necessary to provide extra support at the first girt, in consequence of the necessary omission of diagonal braces, the use of which is impracticable on at least the walking-beam and bull-wheel sides of a standard rig. The vertical bottom panel was an endeavor, first, to provide additional working room on the derrick floor; second, to carry out a bracing system which would make the structure self-contained; and third, to simplify the connections for machinery supports, house framing, etc., so as to reduce the expense of shop work. It was found, however, that the additional

space was not required and in consequence the vertical panels were abandoned in 1911; records indicate that the last derrick made in this fashion was shipped in January, 1912. It should be said, however, that other designs made for derricks as early as 1908 were based on the use of a continuous taper; and, as a matter of fact, the derricks made with the vertical panel were only used in complete rigs.

2. *Types and Sizes of Derricks*

The first short-panel derricks were built for the Carnegie Natural Gas Co. in September, 1908, and were used in drilling gas wells at Waynes-



FIG. 33.—55-FT. SHORT-PANEL STEEL DERRICK, 1909. CARTER OIL CO., SISTERSVILLE, W. VA.

burg and Export, Pa., and at Burton and Hundred, W. Va.; that is, in the gas fields of Washington and Westmoreland Counties, Pa., and Wetzel County, W. Va. They were 80 ft. high and, in connection with machinery supports, made up the first steel rigs ever constructed—as will be mentioned later. It may be noted in passing that while intended for standard cable-tool drilling, the walking-beam side was constructed to

take a calf wheel. This, however, was never used; and after the first erection the calf-wheel posts and braces were abandoned. Three more drilling rigs of the 1909 design, that is, of the pure standard type and somewhat lighter members, were built in the following year and were used by the same company in drilling gas wells in Lorain County, Ohio. These derricks have withstood safely the strains encountered in drilling wells 4,500 ft. deep and while structurally not quite so perfect as those now manufactured have been successful in every way.

In January, 1909, five 55-ft. standard derricks were built for the Carter Oil Co. and used at Sistersville, W. Va. These were short-paneled, tapered from top to bottom, and were used over pumping wells (see Fig. 33).

Otherwise sales in 1909 and 1910 were confined to 80-ft. standard derricks and 72-ft. standard and California derricks; but, as will be noted later, a new type of derrick was introduced, known as the Oklahoma, which has been manufactured in heights of 72 and 80 ft.



FIG. 34.—72-FT. YORKE STEEL DERRICK, 1910. HUGHES SPECIALTY WELL DRILLING CO. BLOWN DOWN AND RE-ERECTED AT STATESVILLE, ALA.

The first 80-ft. California derrick with continuous taper was made in 1911 with Yorke joints and was shipped to Cardenas, Cuba. The same year was also manufactured the first 64-ft. Yorke standard derrick for use in pumping and cleaning old wells, and also the first 64-ft. Woodworth rotary derrick. The latter was sold to the Hughes Specialty Well Drilling Co., Charleston, S. C., and has been used repeatedly in drilling water wells in Georgia, Alabama, South Carolina, and Florida. In addition to the 64-ft. Mr. Hughes also had a 72-ft. Yorke steel derrick in use in his water-well operations. This was blown down by cyclone at Statesville, Ala., straightened out and re-erected by four men in nine days at a cost of \$62. Not a piece was lost; only three pieces had been actually cracked or broken and these were spliced with plates. A wooden derrick in the same storm was totally destroyed (see Fig. 34). Many of the 72-ft. California derricks manufactured in this year went to Casper, Wyo., and were used in the opening up of the new Salt Creek field.

The rotary steel derricks were first offered for sale in July, 1911. Their introduction was due to a first-hand investigation made by the writer of drilling practice in Texas and Louisiana. The first 64-ft. rotary derrick was used for water-well drilling. The second, 72 ft. high, was made in 1912 for the Southern Well Works Co., Chattanooga, Tenn., and shipped to Gillespie & Co., San Fernando, Trinidad, for use in that field. In the same year was made the first 80-ft. California Combination rotary derrick. Four of these were made into complete rigs for the Penn-Mex Fuel Co. and were shipped to Tampico, Mexico, for use in the development of its extensive holdings on the Panuco River. In the same year also were made the first 40-ft. Yorke derricks for the Duquesne Natural Gas Co., Tulsa, Okla. These derricks, while designed primarily for pumping and cleaning, were made of sufficient strength and stiffness to pull casing.

The third field of drilling enterprise was entered in 1913 when two 40-ft. Yorke derricks were sold to the Sullivan Machinery Co. and shipped to Montivideo, Uruguay, for mineral exploration. In the same year this company purchased also a 64-ft. derrick, which was shipped to Cleveland-Cliffs Iron Co., Negaunee, Mich., for mineral exploration on the iron ranges. The derricks were furnished with a different crown block; otherwise there was no essential modification.

The first 80-ft. heavy California derrick was manufactured in 1913 and it is remarkable that the two then made were not used either for oil drilling or for water drilling or for mineral exploration, but were utilized by the Duquesne Light Co. at Neville Island, Pittsburgh, in the support of light cables and other wires across the river during the reconstruction of the Ohio Connecting Bridge of the Pennsylvania Lines West.

Four super-heavy California Combination derricks were made in the same year and used in complete rigs by the Caribbean Petroleum Co. at Maracaibo, Venezuela.

This year is also marked by the introduction of the 59-ft. rotary derricks, the first of which was shipped to the Cruse Syndicate, Port of Spain, Trinidad; the 80-ft. rotary derricks, two of which went to the same company, and others to the Southwestern Gas & Electric Co., Shreveport, La., for use in the Caddo field, and also the 106-ft. heavy Woodworth rotary derricks, the first of which was made for the Oil Well Supply Co. and shipped to the American Trading Co., Buenos Aires, Argentine. Mention should also be made of the 86-ft. heavy California derrick which was first fabricated in this same year as a part of a complete drilling rig.

The 80-ft. derrick is, as a rule, of sufficient height for ordinary cable-tool drilling as practiced in the Appalachian oil and gas fields, in Oklahoma, Wisconsin, Wyoming, etc. Recently, however, the tendency has been in the direction of longer strings of tools and casing, and especially more room is required for the heavier sheaves, hooks, etc., used in deep

drilling. The 86-ft. heavy California derrick, therefore, constructed in 1913, was built on account of the increase in heights required by drilling improvements. It was followed in 1914 by the 86-ft. rotary derrick, the first of which went to F. E. Spence, Semmes, Ala.

The structural steel derrick, as thus developed, has had a gradual but wide distribution. It is to be found in Ohio, West Virginia, Pennsylvania, Michigan, Colorado, Louisiana, Texas, Wyoming, Oklahoma, Montana, Tennessee, Georgia, Florida, Virginia, Peru, Argentine, Uruguay, Trinidad, Venezuela, Cuba, Mexico, Ontario, Alberta, the Orange Free State, the Netherlands, and Burmah. It has been used in drilling oil wells, gas wells, water wells, salt wells, and in mineral exploration. The number of such derricks, while steadily on the increase, has fluctuated somewhat from year to year. An idea of the variety of designs required is gathered from the statistics of manufacture in 1913, in which year their use was not seriously affected by the recent depression in the oil and gas fields. Sales that year included two 40-ft. Yorke standard derricks; one 59-ft. Woodworth rotary; four 64-ft. Woodworth standard; two 72-ft. Woodworth standard; three 72-ft. Yorke standard; three 72-ft. Yorke California; 62 80-ft. Yorke standard; 15 80-ft. Woodworth standard; five 80-ft. Woodworth rotary; five 80-ft. Woodworth California; three 80-ft. heavy Woodworth California; four 80-ft. superheavy Woodworth California, one 86-ft. heavy Woodworth California, and one 106-ft. Woodworth rotary derrick. Sales that year included 30 complete rigs, of which 17 were furnished with structural steel derricks, and 13 were for use with pipe derricks. Derricks were furnished with and without bases. In addition to more or less complete structures, there were manufactured numerous miscellaneous lots of separate bases, house framing, machinery supports, bull wheels, band wheels, calf wheels, walking beams, tops, crown blocks, and girts for pipe derricks, etc.

3. *Derrick Joints*

The long-panel steel derricks were fully spliced at the joints; that is, the structural steel detail designers followed the same method of procedure as was in vogue in the detailing of structural steel towers for water tanks, etc. They put in sufficient bolts and rivets at each joint to develop the full strength of the section, and proceeded in like manner with the diagonal braces and the girts. Some of the joints were made even with both interior and exterior splices. The strength of a derrick is, of course, no greater than the strength of the weakest part; and careful consideration was, therefore, rightly given to this important feature. Experience, however, has indicated that the derrick legs seldom take anything other than direct compression and that in consequence it is safe to proceed with somewhat fewer bolts and rivets than would be considered necessary in structural steel work generally. The tendency, therefore, in the development of the structural steel derrick, as well as in the structural steel

rig, has been to eliminate superfluous bolts and thus to reduce their number to a minimum. The splice should be of sufficient length to prevent lifting of the leg therefrom under an unequally distributed load on the crown block, and there should be sufficient bolts in the girts and braces to resist all probable wind pressures.

In the previous section, reference has been made to different types of derricks by name. These names have been adopted as a matter of commercial convenience and have particular reference to the character of the joint.

a. Woodworth Derricks.—These are covered generally by patent No. 960,474, of June 7, 1910, and have always been made with the standard structural steel splices customary in steel building construction, with the proviso above noted.

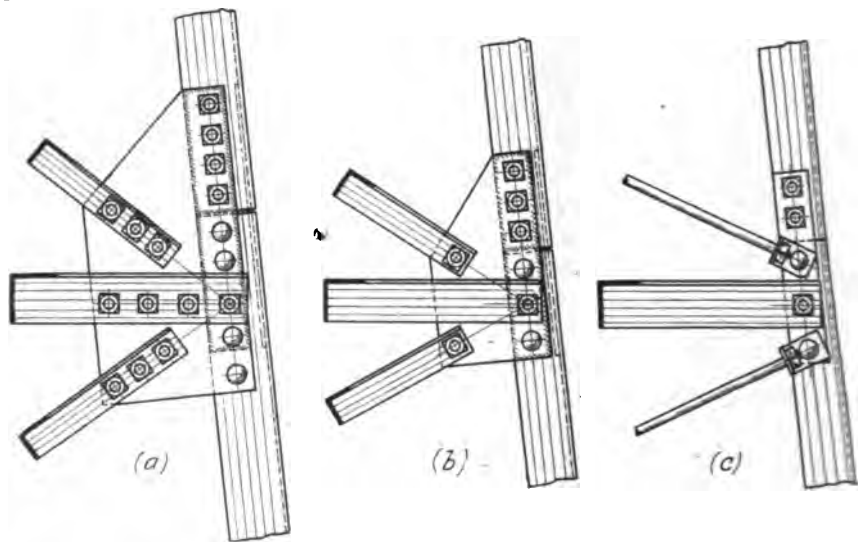


FIG. 35.—WOODWORTH DERRICK JOINTS.

The four short-panel steel derricks built for the Carnegie Natural Gas Co. in September, 1908, and subsequent derricks of the same type, were made with the joint shown in *a*, Fig. 35. The gusset plates at each intersection were riveted to the lower leg sections by eight rivets. There were inside sliding bars to form a pocket for the upper leg section. There were four bolts for each of the girts and three bolts for each of the braces, making with the bolts for the leg a total of 28 to be placed by the rig builders, or a total of 112 at each panel.

In June, 1909, the joint shown in *b*, Fig. 35, was adopted. It will be noted that the inside bar was still retained, and that wide gusset plates were still used but that the number of bolts of each intersection had been reduced to 12.

This style of joint continued in use for about two years until Feb. 25, 1911, when the joint shown in *d*, Fig. 36, was adopted. It will be noted that now the gusset plate was riveted to the girt, that the holes in the legs were all open and that the diagonal braces were run in to take the bolts in the angle legs, the number of which has been reduced to eight. This type of joint was further simplified Aug. 29, 1912, when square-root angles were adopted for splices in the place of plates. This form of joint, shown in *e*, Fig. 36, represents the irreducible minimum. The square-root angle is riveted to the lower leg section. The girts and braces are punched with only one hole in each end, and there are but eight bolts at a joint to be put in and taken out by the rig builders, or a total of 32 at each panel as compared with the 112 which were used in 1908. This joint has

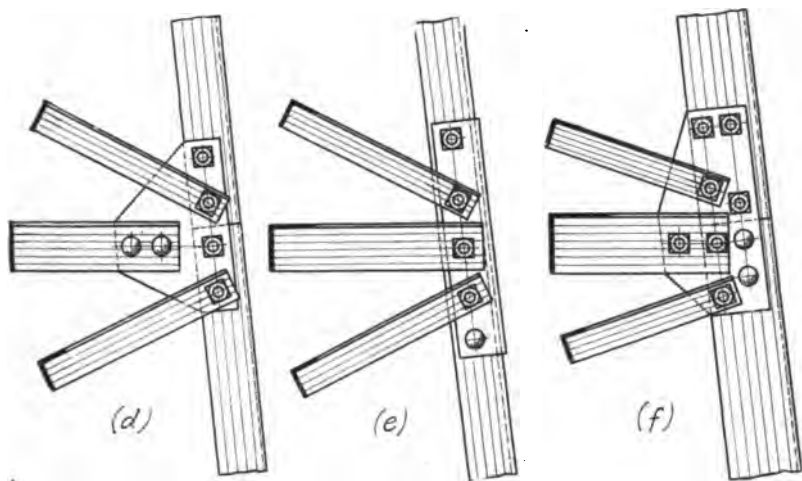


FIG. 36.—WOODWORTH DERRICK JOINTS.

given entire satisfaction, and has been amply adequate to the needs of the structure. No derrick failures have taken place in the Woodworth derricks at the joint, as indeed there have been extremely few for any cause.

The square-root angle joint last referred to is only suitable for the relatively lighter drilling derricks. The largest size square-root angle rolled by any rolling mill in the United States is 4 by 4 in., which can be used on angles with a maximum width of 5 in. The ordinary structural angles are not suitable for derrick joints on account of their inside fillets, which do not permit them to fit snugly against the derrick legs, and in consequence in all the heavy derricks with leg sections 6 in. wide or wider, it is necessary to revert to gusset plates. Fig. 36 shows in *f* the standard joint for heavy structures, consisting, at each intersection, of two plates riveted to the lower leg section by four rivets, and with all the other connections open, a total of 14 at each joint or 56 for each panel. This is the

type of joint used on the 106-ft. steel drilling rig shown at the Panama-Pacific Exposition. It will be noted that while this derrick is figured for a safe working load of 223,000 lb. as compared with the 92,000 lb. computed as safe for the 80-ft. derrick of 1908, there are now just one-half as many bolts, whereas by the older method of computation two and a half times that many would not have appeared unreasonable. The leg sections are not faced, but the holes in the upper section are spaced on a slightly larger spacing than the holes in the gusset plates or square-root angles, and in consequence, when erected, the two sections butt smoothly and tightly together. The legs are cut by the mills with extreme accuracy, and in consequence the $\frac{1}{16}$ -in. play in the bolt holes is amply sufficient to insure a solid bearing, and this solid bearing is the thing that is now relied upon, rather than the full splicing of the joints with bolts in shear, to transmit the load from the crown block to the derrick base.

b. Woodworth Oklahoma Derricks.—In 1909 the Oklahoma Iron Works secured ten derricks in heights of 55 and 68 ft., from Joseph T. Ryerson & Son in Chicago, and endeavored to introduce them in that district. So far as the writer can recall, these derricks were of rather crude construction, the girts, made of small channels, being extremely flexible, and the joints inadequate. An investigation was made of the needs of that field and in that year a very light type of derrick was placed on the market in the expectation that it would prove satisfactory in the drilling of shallow wells and in pumping and cleaning old wells.

The type of construction at the joint is shown in *c*, Fig. 35. Splices were made by means of short bars riveted to the lower leg section, above which were riveted lug angles to take the diagonal braces, which were made of round rods. These, however, were without turnbuckles. They were made with long threads and were adjusted to proper tension by the nuts at their ends. The use of these threaded rods was adopted in the first instance to reduce weight and in the second instance because drillers had not given any favorable attention to turnbuckles.

These derricks were offered for sale in heights of 55, 64, 72, and 80 ft. Seven 72-ft. derricks and six 80-ft. derricks were placed in service in 1910 and 1911 in Ohio and in Oklahoma. The use of these derricks did not commend itself to the operators, and their manufacture has, in consequence, been discontinued. They are, however, believed to be suitable for light work; but the decrease in weight is somewhat offset by increase in manufacturing cost.

c. Yorke Derricks.—In July, 1908, the writer made the acquaintance of Patrick Yorke, of Washington, Pa., a drilling contractor of some 30 years' experience in the actual production of oil, who had made a number of improvements in the art of drilling. He had patented the bolted wooden derrick shown in patents No. 852,486, of May 7, 1907, and No. 877,624, of Jan. 28, 1908, to which were added subsequently No. 898,027,

of Sept. 6, 1908, and No. 946,815, of Jan. 18, 1910. The essential feature of these patents so far as they applied to derrick construction was the use of slots formed on the upper and lower edges of connection plates and braces, instead of round holes, the girts and braces being adapted to be clamped together at the different sections by bolts.

The portable steel derricks manufactured in accordance with these ideas have been known as the Yorke portable steel derricks, advertised as the only derricks which can be dismantled without removing a bolt—which is strictly true of the main joints only.

Inasmuch as the strength of the joints in a structure of this fashion is determined not by the shearing resistance of the bolts but by the friction of the gusset plates against the main members, when the nuts have

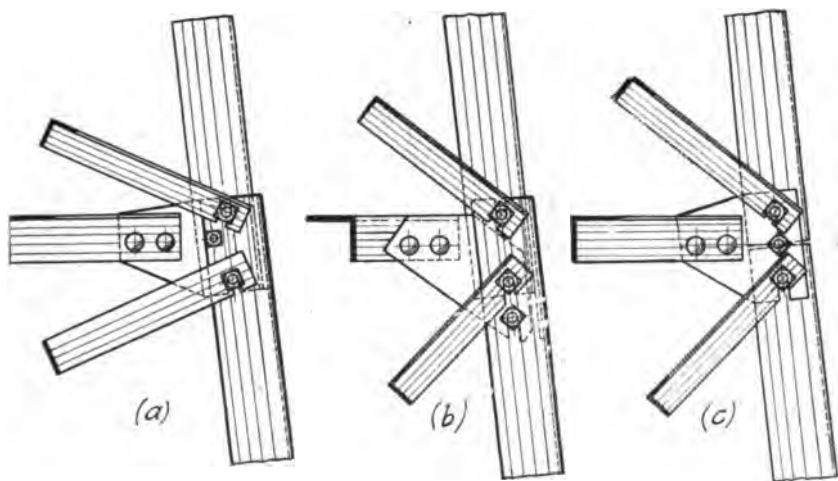


FIG. 37.—YORKE DERRICK JOINTS.

been drawn up tightly, the idea did not appeal very strongly at first glance to men experienced in the fabrication of structural steel. The Carnegie Steel Co., however, undertook the manufacture of the Yorke derrick under arrangements with Mr. Yorke and continued to manufacture it down to the end of 1913, when the association ceased by mutual consent. Mr. Yorke's experience in the actual drilling of wells combined with the experience of the Carnegie Steel Co. in the manufacture of structural steel resulted in the development of a drilling structure well suited for the lighter kinds of work done in standard-rig drilling practice.

The first Yorke derrick manufactured by the Carnegie Steel Co. was shipped Jan. 8, 1909, to the Manufacturers Light & Heat Co. for use near Waynesburg, Pa. It had no steel crown block or steel base, and the ladder was made of wood. Steel bases, ladders, and crown blocks were, however, added in the development of the derrick, and other improve-

ments were made from time to time, so that their manufacture was ultimately standardized and they were used interchangeably with the Woodworth derricks and made from the same shop detail drawings so as to be interchangeable with them in the construction of complete rigs.

In the first derricks the joints were made as shown in *a*, Fig. 37. The connection plates were riveted to the girts and the derrick legs were overlapped on each other the full depth of the gusset, to form the splices. In May, 1909, details were modified as shown in *b*, Fig. 37, so as to provide a longer splice with a somewhat simpler method of erection, by reason of the elimination of the countersunk bolt at the center of the gusset plate.

In August of that year, the detail shown in *c*, Fig. 37, was adopted.

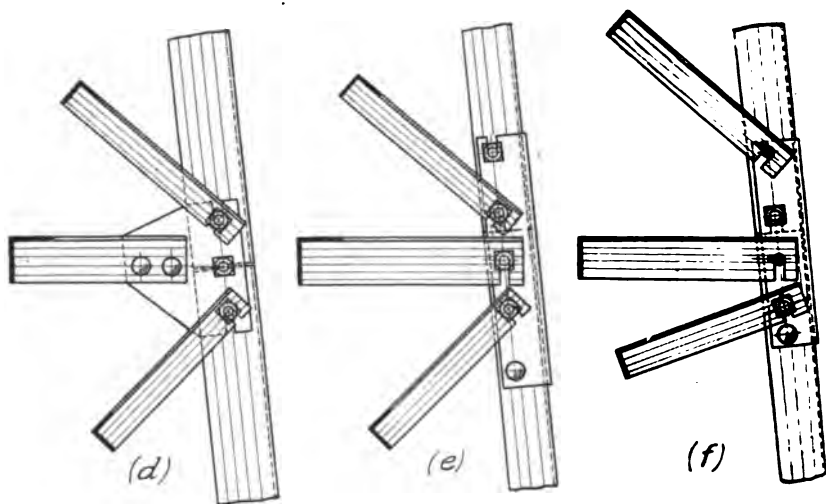


FIG. 38.—YORKE DERRICK JOINTS.

The legs were made abutting instead of overlapping, with the girt riveted to the gusset plate.

This method of construction was followed for two years, during which time a number of them were placed in service. There were, however, one or two failures by reason of the pulling out of the derrick legs from the splices, under the influence of unequally distributed loading on the crown block. The difficulty was finally traced to the fact that the splices were of insufficient length and in consequence a modification was made as shown in *d*, Fig. 38, which is the same as *c*, Fig. 37, except that the splice was made longer. The slots in the diagonals were now made at right angles instead of vertical.

This solved that problem and there were no further difficulties on that score; but when the Woodworth derrick joints were changed from gusset plates to square-root angles in August, 1912, the Yorke derrick joints were modified in the same manner and in consequence were made in

accordance with *e*, Fig. 38, and so continued to be made until the end of 1913. On Mar. 24, 1914, Mr. Yorke was granted patent No. 1,090,950 covering further improvements in metallic derricks for use in oil, gas, salt, and artesian wells, although equally applicable for mining shafts, hoists, and other towers generally, the parts of which were said to be so constructed and arranged that they might be quickly assembled and disassembled without putting in or taking out a single bolt. The patent lock-bolt construction in its latest form is shown in *f*, Fig. 38, and I understand that a number of them have been manufactured by the O'Brien Steel Construction Co., Washington, Pa., and placed in service in the Appalachian oil and gas fields.

d. The Foukes Joint.—While the Yorke and Woodworth drilling derricks have given good service, and their erection has been simplified and cheapened by a number of improvements, introduced as already stated, the derricks still remain subject to the usual methods of shop

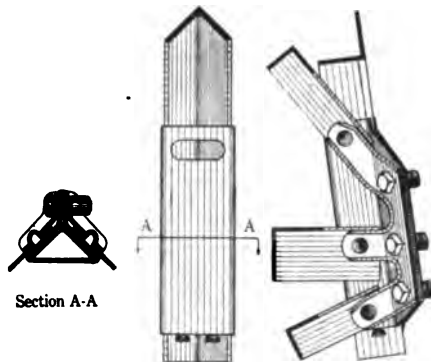


FIG. 39.—FOUKES DERRICK JOINT.

manufacture, whereby wooden templates are made, the holes marked off in accordance therewith and punched. To do away with these elements of shop cost, as well as to simplify the work of erection, various expedients have been conceived, none of which have come to actual fruition with the exception of that now to be noted.

The Foukes joint is the invention of Philip Foukes, Superintendent of the Upper Union Mills fitting shop of the Carnegie Steel Co., and his method of construction is shown in Fig. 39. It consists essentially of a forged-steel plate formed to shape under dies, to take the derrick legs, with an outstanding wing to receive the girts and braces. The derrick legs do not require any punching, except that a small button is formed on their upper ends just below the clamp, which serves as a temporary support during the process of erection. When the derrick is erected the leg section is hoisted into position with the clamp in place, girts and braces are inserted, and, when all connections at the joint have been made, the

bolts are tightened, and, by reason of the spring action in the plates, not only are they rigidly held in place but also the clamp is firmly attached to the legs.

This joint, tested previously in the shop, was first introduced in an experimental way by the Carnegie Natural Gas Co., in 1914. Seven derricks for drillers and pumpers, 80 ft. and 64 ft. in height, have been erected and have given complete satisfaction.

VIII. STRUCTURAL STEEL DRILLING RIGS

By the substitution of structural steel for wood in derrick construction little reduction is made in weight, for the reason that there is little excess of material in the wooden derricks—a result of long experience in their construction. This is not the case, however, with such pieces as samson posts, jack posts, foundation sills and other heavier portions of the sub-structure. With steel, it is practicable to adjust very nicely to exact requirements the strength, and with it the weight, of members. While some economies have been effected by the use of steel derricks, much more has been gained by the substitution of steel for wood throughout the entire drilling structure, so that, while a steel derrick of equivalent height and strength to a wooden one would weigh approximately the same, a complete steel drilling rig will weigh only about three-fourths as much as the corresponding structure built in wood. This reduction in weight not only reduces the cost but is an important consideration in a structure intended to be semi-portable. For this reason, when the introduction of steel drilling structures was commercially undertaken, endeavors were made to carry the substitution of steel for wood to the utmost extent.

1. *Rigs Wholly or Chiefly of Structural Steel*

It has been noted that the two 72-ft. California steel derricks shipped to South America in 1908 were complete with base, steel walking beam, machinery supports, house framing, and ladder. This was good so far as it went. The next step was to build a rig complete with derrick, crown block, ladders, base, machinery supports, samson post, jack posts, knuckle posts, walking beam, bull wheel, band wheel, house framing, etc., all of steel, and to cover it with corrugated steel roofing and siding. This step was taken in 1908 when four 80-ft. rigs were built for the Carnegie Natural Gas Co. for use in Washington and Westmoreland Counties, Pa., and Wetzel County, W. Va. The construction of the derricks for these rigs has already been referred to. The working parts were built entirely of structural steel, with the exception of the crown pulleys and sand reel, which, having been made already in metal, it was not necessary to re-design. The house framing, however, was continued out only so far as

the sand reel, the remainder of the belt house and the engine house being of wood (see Fig. 40).

In the same year a complete design was worked out for a 55-ft. rig to be used in cleaning and pumping old wells and for shallow drilling. Rigs of this sort have not been built. The wheels and walking beam of the standard rig have, however, been used with 64-ft. derricks.

Three 80-ft. Woodworth drilling rigs were built for the Carnegie



FIG. 40.—80-FT. STEEL DRILLING RIG, 1908. CARNEGIE NATURAL GAS CO., EXPORT, PA.

Natural Gas Co. in 1909 and were first erected at Avon, Ohio, and Lumberport, W. Va., for the drilling of gas wells. It is usually impracticable to follow up the records of drilling equipment manufactured and shipped broadcast to the ends of the earth. In the case of these rigs, however, records of the Carnegie Natural Gas Co. were available. By the end of 1910, one of the 80-ft. rigs made in 1908 had drilled five wells with a total depth of 13,901 ft. and up to the end of 1912 one of the 1909 rigs erected at Avon, Ohio, had drilled six wells with a total depth of

17,808 ft. One of these wells, C. M. Green No. 1, Avon Township, Lorain County, Ohio, reached the depth of 4,440 ft.

Six 72-ft. California rigs were made in 1909 and shipped to the Burmah Oil Co., Ltd., at Rangoon. These rigs, however, while fitted up complete with derricks, machinery supports, house framing, walking beam and samson post, and metal sand reel, utilized wooden bull, calf, and band wheels, which is an instance of what has happened all through the course of the development. The equipment, while designed primarily to be used as a unit, is interchangeable with other types of material, such as wood and pipe. This would not be the case had the designer not deliberately



FIG. 41.—72-FT. STANDARD STEEL DRILLING RIG. CARNEGIE NATURAL GAS CO., MUNHALL, PA. RECORD, FIVE WELLS.

chosen to follow the well-recognized lines of oil-well structures and to depart therefrom only for very good and sufficient considerations.

The first standard combination rigs were manufactured in 1912, 80 ft. high, and were shipped Apr. 25. The first went to I. N. Knapp, Houma, La., and the second to Icaos Development Co., Trinidad. In the Knapp rig, it may be noted, the engine was placed between the draw works and the band wheel, both of which it drove by sprocket drive. Mr. Knapp drilled three wells with this rig in the search for oil in Terre Bonne Parish, and after the endeavor was definitely abandoned on account of geological conditions, the rig was sold to the Southwestern Gas & Electric Co., of Shreveport, La., at a very handsome figure with previous use

considered, and has doubtless drilled a number of wells since. The Icacos rig drilled a fine oil well at its first location in 62 ft. of sand, at a depth of 2,859 ft., the first 1,032 ft. of which was drilled with cable tools and the remainder with rotary, the diameter of the hole being 16 in. at the top and 8 in. at the bottom.

The first 80-ft. California combination rigs were built the same year, three of them going to the Penn-Mex Fuel Co. at Tampico. The first 80-ft. heavy California combination rigs were made in 1913 for the Carib-

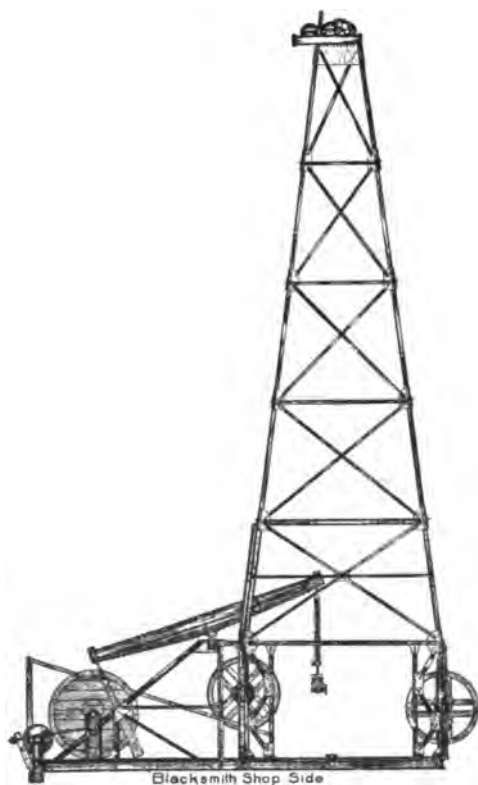


FIG. 42.—72-FT. WOODWORTH CALIFORNIA RIG. THE 1904 STRUCTURE.

bean Petroleum Co., and shipped to Maracaibo, Venezuela. The first 86-ft. heavy California rig was made in 1913 on order of the Oil Well Supply Co. for the Calgary Petroleum Products Co., Calgary, Alberta, Canada.

It has been impracticable to follow up the record of these rigs. They have been built on the same general lines with California double-drum sand reels located on the walk side. In 1913, however, a rig was built for the Hope Natural Gas Co. for use in West Virginia to take a very long special sand reel set on a bevel. This, of course, necessitated a

thorough revision of the general design. While some drillers have objected somewhat to the use of the straight-line reel set on the walk side, experience has indicated that drilling rigs can be built much more compactly in that way; and when standard rig drillers become accustomed

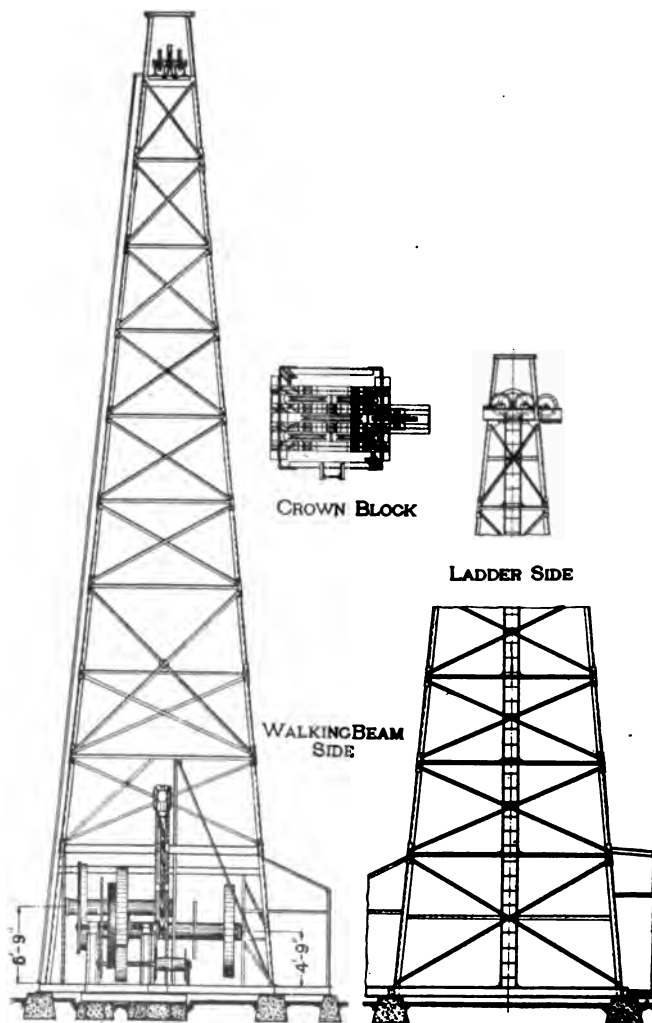


FIG. 43A.—80-FT. WOODWORTH CALIFORNIA DRILLING RIG. THE COMPLETE MODERN STRUCTURE.

to the modification by practice, the construction is found to be very satisfactory; just as in the case of the wooden California rig, in which the use of an iron sand reel set on the walk side has been standard practice for many years (see Figs. 43A and 43B).

Practice in the construction of combination rigs is not yet absolutely

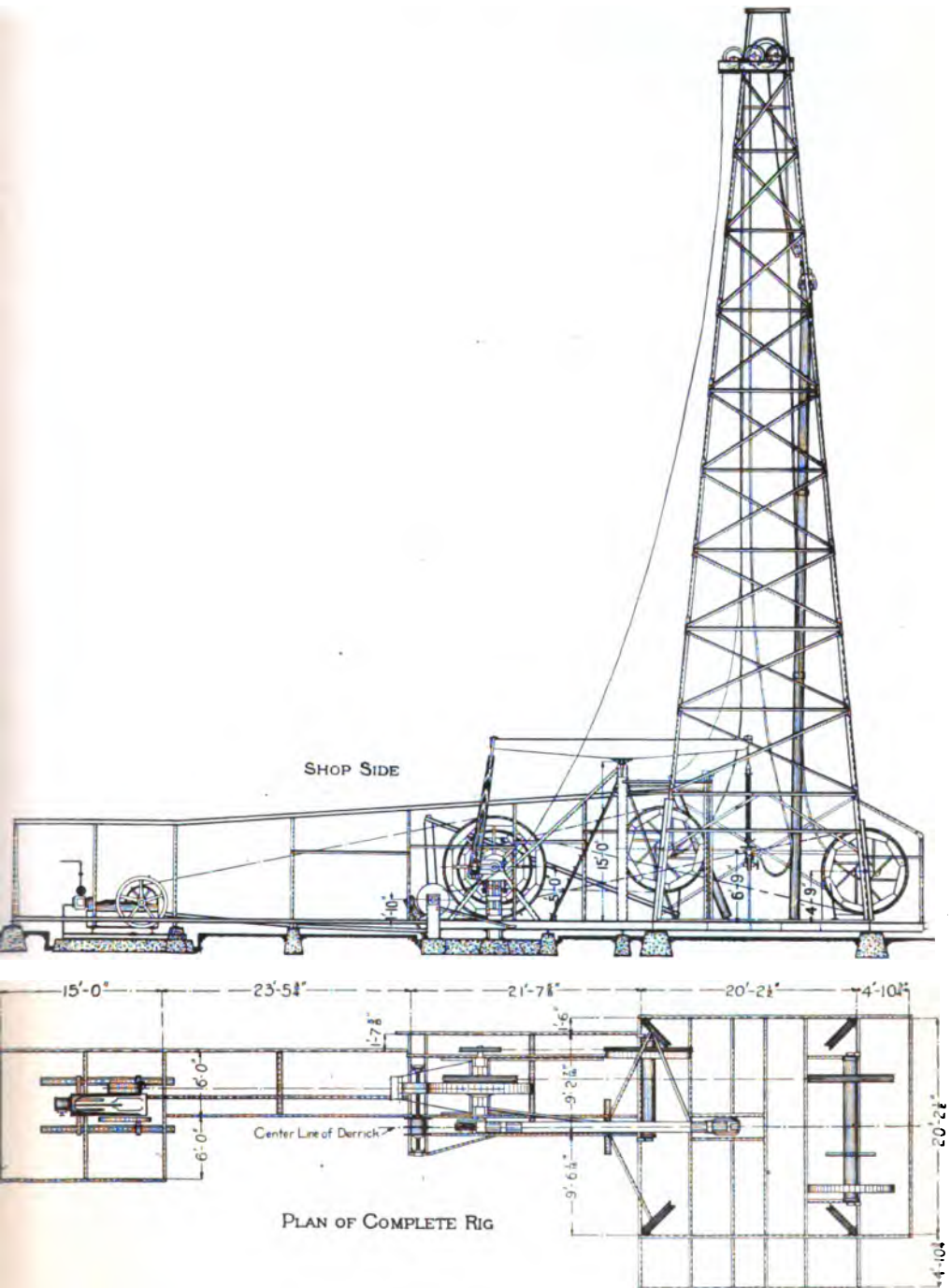


FIG. 43B.—80-FT. WOODWORTH CALIFORNIA DRILLING RIG. THE COMPLETE MODERN STRUCTURE.

standardized. The rigs already referred to have been built for use with one engine, from whose pulley power is transmitted to the band wheel by belt, and from the band wheel to the bull wheel or calf wheel in the usual manner, and to the draw works located on the walking-beam side by supplementary sprocket on the band-wheel shaft. The 106-ft. heavy California combination drilling rig (Fig. 44) now on exhibition at the

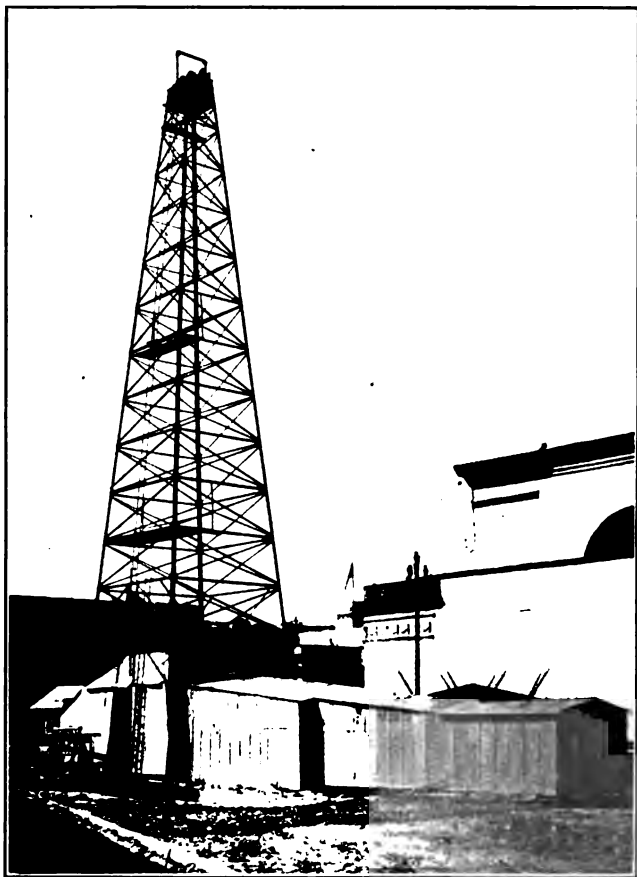


FIG. 44.—106-FT. CALIFORNIA COMBINATION DRILLING RIG. PANAMA-PACIFIC INTERNATIONAL EXPOSITION, 1915.

Panama-Pacific International Exposition may be considered as representing the culmination in this line of development and also as the most complete drilling machine ever constructed. It is designed for use with two engines, one to drive the rotary and the other for use with cable tools. The walking beam is mounted on revolving center irons, and can thus be swung out of the way of the drill pipes when the rotary is in use. As a consequence, the most rapid interchange of drilling tools can be effected,

and drilling proceeds with rotary or cable at the option of the operator. All the material is structural steel except the floor platforms, the cants for band, bull, and calf wheels and the nailing strips to which the corrugated iron covering is attached. The bull wheel is 8 ft. in diameter with an 18-in. O. D. pipe shaft, 16 ft. long over gudgeons; the calf wheel is 7.5 ft. in diameter, 16-in. pipe shaft, 90-in. sprocket; the band wheel is 11 ft. in diameter, 12-in. face, with 7-ft. tug wheel, double drive; the band-wheel shaft is 6 in. in diameter with 42-in. calf-wheel sprocket; the sand reel is a California double drum, with 42-in. pulley and 40-in. flanges; the walking beam 12 by 24 in., 26 ft. long between centers of bearings, equipped with cable protector and spring pitman box. The pitman is of wood. This could be made of steel, and it is understood that steel pitmans (Smith and Hess pattern) have come into recent use on wooden rigs in the California field. They have also been used experimentally in West Virginia gas-well drilling. The steel foundation of this rig is set on concrete blocks to represent the most recent and best practice. The total weight of the rig, not including wooden floor, platforms, and nailing strips, is 81,310 lb. It will sustain an ordinary working load of 223,000 lb. on the usual factor of safety of four. It would probably require a direct pull of at least twice this amount before serious injury were done to the structure.

Structural steel drilling rigs adapted to the Canadian method of drilling with steel derricks 56 ft. high are shown in the 1909 catalogue of the Deep Well Tool & Boring Co., Ltd., St. Albans, England. They appear also in the 1912 catalogue of the Oil Well Engineering Co., Manchester and London, England. Steel drilling rigs adapted to the Galician method of drilling are advertised by the Galician-Carpathian Petroleum Corporation. The writer has no specific information as to how far these rigs have come into use or details of their construction.

2. Pipe Rigs

As already noted, the principle of interchangeability has been kept in mind in this development. Parts of the steel equipment may be, if desired, of wood. Such use does not concern the manufacturer. He is interested, however, when structural steel drilling equipment is to be used with pipe derricks, since in that case he must work out the shop details necessary to insure satisfactory operation.

The first structural steel rigs for use with pipe derricks were made in 1909 for the South Penn Oil Co. Eleven of them were used in that year in the Shinnston district. These were not complete. What the structural steel man furnished was the derrick foundations, machinery supports, house framing, corrugated sheeting, walking beams, and bull wheels. Pipe derricks were made by the South Penn Oil Co. in its own shops, and

the band wheels, sand reels, band-wheel shafts, etc., were taken from its own stock. Numbers of these rigs, more or less complete, for use with pipe derricks, have been built since that date, chiefly in conjunction with Lee C. Moore & Co., who are the manufacturers of the Neill pipe derrick, as already noted. The South Penn Oil Co. has some three dozen of these



FIG. 45.—STEEL-PIPE DRILLING RIG FOR CABLE DRILLING. INGERSOLL-RAND CO., NEW YORK, N. Y.

rigs in use in its extensive drilling work in West Virginia. The very heaviest so far built, however, were two 95-ft. California combination rigs constructed in 1914 for Lee C. Moore & Co. and shipped to the Anglo-Mexican Petroleum Products Co., Tampico, Mexico. These rigs weighed 53,000 lb. each, not including the pipe derrick.

3. *Parts of Rigs*

Just as improvements have been introduced in the manufacture of steel derricks from time to time, generally in the direction of the simplification of details and the reduction of weight, so also have improvements been made in the separate parts of the drilling rig, which are worthy of notice as bearing on the present state of the art and the methods by which it was reached. Some of these are covered by letters patent: No. 932,081

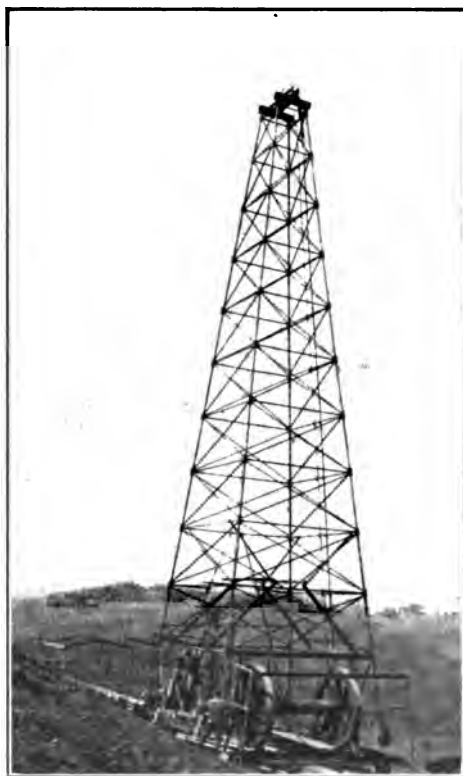


FIG. 46.—STEEL-PIPE DRILLING RIG FOR OIL-WELL DRILLING. SOUTH PENN OIL CO., BLUE CREEK, W. VA.

and No. 1,090,184 on bull wheels, No. 1,020,069 on band-wheel construction and, No. 960,474 on the general assembling and make-up of a complete drilling rig.

a. Crown Blocks.—The first two-pulley crown blocks were constructed of steel channels with babbitted top and bottom bearings, and used on derricks made in 1908 and 1909. The users, however, did not seem to require top bearings and so these were discontinued. As a matter of fact, a great many still prefer not to have any metal bearings whatever,

but to use pieces of wood. The first crown blocks were without any protection against cable wear. The present and most approved method in the construction of two-pulley crown blocks for standard drilling is to make them of steel beams with babbitted bottom bearings and to line the top and bottom flanges with wood, in order (1) to prevent the drilling cables from coming in contact with the sharp edges of the beams at the bottom; and (2) to provide a proper wearing surface at the top where sheaves are hung below the crown block for pulling the casing (see Fig. 47).

The four and six pulley California crown blocks are likewise made of beams and protected by wood; but inasmuch as these blocks are used with wire lines, the most recent tendency is to omit wood, which is likewise done in the case of the five-pulley block used on rotary derricks. The 106-ft. steel drilling rig at the Panama-Pacific International Exposition is furnished with a seven-pulley crown block with babbitted bottom bearings, and with top steel bent plate covers fitted with grease cups.



FIG. 47.—SIX-PULLEY CALIFORNIA CROWN BLOCK.

The steel crown block has been in use many years on wooden derricks. The first known to the writer were used in Texas on rotary derricks in 1909. Very heavy blocks fitted with special bearings, self-oiling devices, etc., have within the last year or two come into use in the California fields, and are made by the larger dealers in oil-well supplies in their own shops. Some of them are made to permit adjustment in the position of the pulleys at almost any desired location, extra holes being provided for that purpose. Such are used by the Standard Oil Co. of California in its rotary drilling.

b. Bull Wheels and Calf Wheels.—Steel drilling rigs are characterized by the use of bull wheels built with channel spokes and wood-filled plate rims attached to a pipe shaft. The spokes of the first bull wheels made in 1908 were constructed of 6-in. I-beams connected to the shaft through forged angle flanges and set radially, and the rims were made of steel channels with the flanges turned outward and with wooden cants laid in and bolted thereto in the line of their depth. The gudgeons were of bowl type and were screwed on the outside of the pipe with stand-

ard threading with, in addition, four set screws to prevent slipping (see Fig. 48).

The wheels are now made with 16 or 18 in. O. D. pipe shafts and with inside gudgeons bolted to the pipe. The spokes are made of steel channels, bolted tangentially to the shaft, which method does away with the forging required in making the angle flanges. The brake wheel has a 9-in. face and is made with a plate rim riveted to the channel spokes and carrying wooden fillers similar to the standard wooden cants. The tug wheel is made in the same way, except, of course, that the cants are grooved for double tug, and provision is made for the attachment of a wooden dog for drawing the rope on and off. The hand holds for turning the wheel are made of gas pipe bolted by $\frac{5}{8}$ -in. bolts through the entire thickness of the wooden cants and the plate rim.



FIG. 48.—8-FT. STEEL BULL WHEEL, 1908. RIMS ARE NOW MADE AS SHOWN IN FIG. 50.

The spool is made of steel angles bolted to the pipe shaft and connected to each other by gusset plates so as to form practically a single piece. Protection of the drilling cable from the sharp edges of the steel is given by two coils of rope wrapped around the shaft next to the tug wheel (as is often done in a wood wheel) or by wooden strips bolted to the spokes, the latter being the preferable manner. Otherwise, the wheel is ready for use as it comes from the maker.

Bull wheels are usually shipped in four pieces, the two wheels, spool, and shaft being separate. Patent No. 1,090,184 discloses a sectional wheel designed with a view to facilitate shipment. No wheels made in accordance therewith have yet been constructed. Bull wheels with metal tug rims, however, have been made for experimental purposes and their service is now under observation.

Some other types of bull wheels may be noted in passing. Ross and Seeley, Hollywood, Cal., patented Jan. 10, 1911, No. 981,128, iron bull-

wheel and calf-wheel shafts, made of iron pipe with outside screwed gudgeons. The rims and spokes are of wood and connected to the shafting by heavy castings (hubs). The calf-wheel shafts are lagged with rope to make the shaft the proper size and form a cushion for the wire casing lines. The bull wheels are not lagged, and the spools are constructed by the use of heavy wood friction blocks made of 6 by 16 in. timbers. Special bearing boxes are provided for attachment to the wooden bull and calf wheel posts. A number of these have come into use in California.

Within the last year or two, the California fields have seen the introduction of the Keck and Hively bull wheels and calf wheels. These consist of pipe shafts with bowl gudgeons and square wooden friction blocks bolted to the shaft for the attachment of the wooden spokes necessary to complete the rim. Otherwise, it is a device for utilizing on

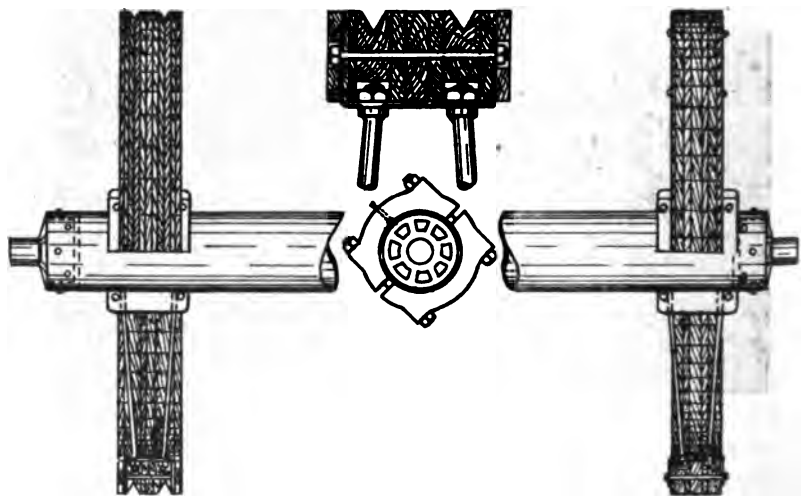


FIG. 49.—MOORE SECTIONAL BULL WHEEL.

a metal shaft regular wooden cants and spokes customary in the construction of wooden rigs.

Moore's sectional steel bull wheels, however, a few of which have been introduced in the Eastern oil and gas fields, are designed to provide an all-metal wheel with the exception of the rim. They utilize 16-in. O. D. pipe, inside cast-steel gudgeons, sectional hub castings, and tangent screwed spokes, carrying at their circumference metal plates on which the cants are set, as distinguished from the method employed in the Woodworth wheels, in which the cants are set within the rim (see Fig. 49).

Woodworth calf wheels have been made in the same manner as the bull wheels and indeed with the same size of pipe and same gudgeons, with a view to the most economic manufacture. The first wheels made in 1909 were without flanges; but flanges are employed in most recent construc-

tion. The wheels have been made with wooden grooved rims for manila rope drive, with iron rims for manila rope or wire line drive and with sprocket rims for sprocket drive.

c. Band Wheels.—The band wheels furnished with the rigs of 1908 were 11 ft. in diameter, with I-beam spokes, channel rims, and the regular cast band-wheel flanges keyed on a $4\frac{1}{2}$ -in. band-wheel shaft. The flanges of the channel rim were turned inward, and a canvas belt was fastened to the outside, so as to provide proper friction against the sand-reel pulley. The cast flanges proved too weak for this service, and on subsequent wheels were replaced with wrought-steel hubs built up of successive plates. The flanges on the channel rim were then turned outward, and filled with wooden cants similar to those first employed on the steel bull wheel, which

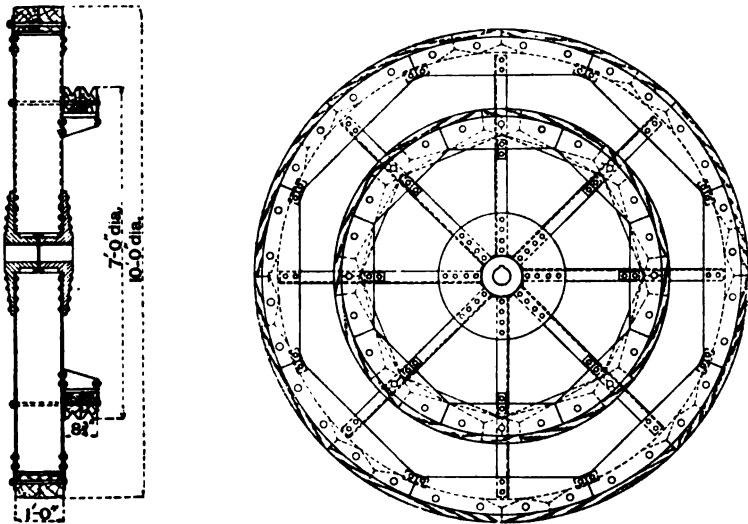


FIG. 50.—10-FT. STEEL BAND WHEEL, 1911. THE RIMS OF THE STEEL BULL WHEELS ARE NOW MADE IN THE SAME MANNER.

gave better friction against the iron sand-reel pulley. In both cases the tug wheel was made of a channel rim, filled with wooden cants and connected with the spokes of the band wheel by angle brackets. The present type of steel band wheel was adopted in 1911. The hubs are made of two solid rolled-steel flanges, abutting together at the center of the wheel, with channel spokes and plate rims, within which are secured the wooden cants. The tug rim is made of grooved cants, faced with a plate and supported on brackets. A hub of this character was first used on a band wheel by Patrick Yorke early in 1911, but the construction of the present wheel deviates very widely from that original form; and the manufacture of the hubs by rolling instead of casting is only possible on the mill operated by the Carnegie Steel Co. at Homestead Steel Works in

the manufacture of a very great variety of annular sections such as car wheels, crane-track wheels, locomotive pistons, etc. The band wheel was first made 11 ft. in diameter, but experience in their use indicates that a 10-ft. wheel on a steel rig is as effective a prime mover as an 11-ft. wheel on a wooden rig (see Fig. 50).

d. Walking Beam.—For some reason or other, designers of steel walking beams have considered it their duty to depart rather widely from the ordinary wooden construction which so admirably served its purpose

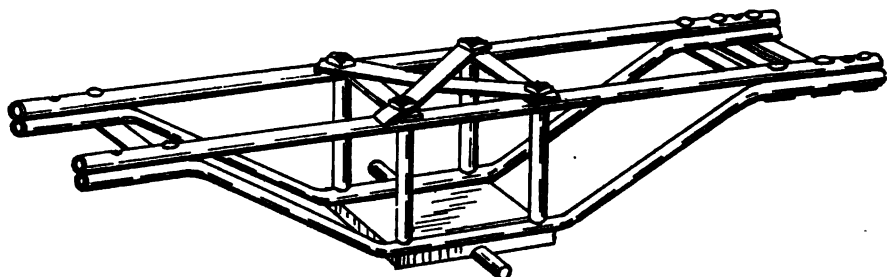


FIG. 51.—SHANNON PIPE WALKING BEAM.

all these years. This has been done by the use of truss rods over the top of a light wooden beam, as was formerly done in the construction of underframes for steam-railway freight cars. Such a reinforced walking beam appears even so late as the 1914 edition of the catalogue of a large dealer in-oil-well supplies. The attempt was next made to use pipe. Patent No. 914,608, issued to Hezekiah Shannon, of Cyclone, Pa., Mar. 9, 1909, shows an all-metal walking beam made of bent pipe and steel bars. The writer does not know of any actual use of a beam built in accordance

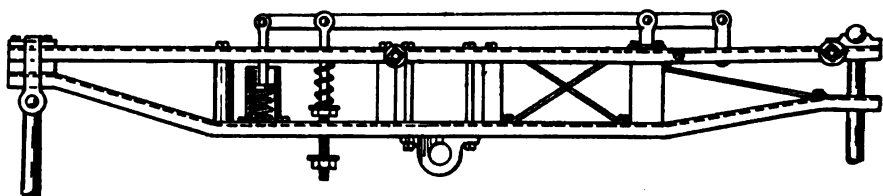


FIG. 52.—LONGTINE STRUCTURAL WALKING BEAM.

with this particular patent but has seen similar beams in actual use, one in the McDonald oil field in the year 1909 (see Fig. 51).

He also saw in February, this year, at Brea, Cal., a very ingenious affair built up with toggle joints, springs, screw adjustments, etc., by the Los Angeles factory of the Oil Well Supply Co. for one of their customers. He made no sketch of the structure at the time, but the beam was similar, if not identical, with that patented by Robert B. Longtine of Taft, Cal., No. 1,009,537, Nov. 21, 1911 (see Fig. 52).

The beams built by the writer have been exact equivalents of the standard wooden walking beams. They are made in box-girder fashion of steel plates and angles, customary tapered shape, and with corresponding notches for temper screws. The first beams were made with wooden bearings and without any cable protector on the temper-screw end. The improvements introduced since 1908 are chiefly a bent pipe at the temper-screw end for a cable protector, and babbitted cast bearings for the pitman. The beams are designed for support on a standard cast-iron saddle, such as is carried in stock by dealers in oil-well supplies, and are interchangeable on wooden or steel rigs. Combination rigs have been equipped with the modern revolving center irons, as already noted.

A great number of steel walking beams are in use. There has been but one real objection to them, and that is that their stiffness made it more difficult for the driller to determine exactly what was going on at the bottom of the well. This stiffness is in part due to the fact that while the steel beams are designed to be of the exact theoretical strength of the standard wooden beams, for which they are substitutes and are very much lighter, yet owing to the greater homogeneity of steel they are really much stronger than the wooden beams. Some three years ago the Oil Well Supply Co., at the instance of the South Penn Oil Co., devised a special spring pitman box, which has been utilized on steel beams and on wooden beams as well.

Steel walking beams with single webs, built up plate-girder fashion (I-shape), have also been devised, but have not come into anything more than limited use, the preference of the driller for what he has been accustomed to use being somewhat greater than his desire to save slight additional expense.

In the first samson-post design, made by the writer as far back as 1904, he contemplated the support of the steel walking beam on a pin driven through its center exactly as is done in other operating mechanism, and this method of construction is advocated in recent publications. It did not come into use, however, until the present year, when it was employed on a heavy California drilling rig built for C. W. Washburne, Angola, West Africa.

CONCLUSION

While it is not at all likely that the steel drilling rig will replace the ordinary wooden structure to anything more than a limited extent, particularly in the drilling of wells for oil, a sufficient number have been put into use to justify conclusions as to their advantages, based not on theoretical considerations but on actual experience.

The steel shop methods insure that if the structure goes together at all, it will go together with a precision not possible in wooden construction. If the sills are level, the structure will be plumb and square without

any particular attention on the part of the rig builder to make it so, and simply for the reason that it has been made so in the shop. It is a question, therefore, whether the steel structure should not rather be called a drilling machine than a rig, and indeed that method of designation was used in the first pamphlets issued by the Carnegie Steel Co. It was withdrawn for the reason that the word "rig," while designating in ordinary parlance something more or less crude, is a standard term without reproach in drilling practice.

The necessary consequence of this precision in manufacture is seen in the smoothness with which all parts of the mechanism work. The structure is stiff and rigid. It does not give like a wooden rig and in consequence there is no lost motion in operation. The thing which was particularly noted by P. Keegan, who operated a 72-ft. drilling rig at Ladybrand, Orange Free State, South Africa, in 1910 on the property of the South African Oil Co., was that he only needed to carry 80 lb. of steam on his boiler. His experience in the reduction of boiler load has been amply corroborated by others. The difference between the pressure necessary on a Woodworth standard drilling rig and a crude wooden structure is due to the smoothness of operation resulting from precision in manufacture and alignment.

In 1892 when the Oil Well Supply Co. first described the steel derrick in its catalogue, the theoretical statement was made that while the first cost would be more than that of a wooden rig, it would be amply compensated for after the completion of the third well. It is reported that some of the steel derricks have been used 20 or more times. Records have been kept of the use of particular steel drilling rigs and they show more than a half a dozen wells each. This argument is, therefore, a good one. At present, a steel drilling rig in the Eastern oil and gas fields will cost, erected, about $1\frac{3}{4}$ times as much as a wooden one, but it represents an investment of indefinite use and practical indestructibility. Assuming that a wooden rig will drill two wells, and a steel rig, say, only four or five, the investment in the latter, capitalized at the usual rate of interest, will yield larger returns, by reason of its small cost of maintenance. This is, of course, particularly true in the case of the very high and powerful structures which deep well drilling has recently made necessary.

In addition to these considerations may be mentioned briefly the fireproof character of the structure, its small wind exposure, and the ease of its erection. Not long since, a steel rig was struck by lightning, which only destroyed a few wooden planks at the top and some flooring. Another rig suffered not long since from a gas explosion which destroyed the cants from the bull wheel and a portion of the wooden floor, but did absolutely no other damage. The surface of the derrick proper exposed to wind is not more than half as great as that of the wooden structure of the same height, strength, and stiffness.

The Serviceability of Gurley Surveying Instruments



No. 28 Transit
Weight,
10½ lbs.

Means not only long service, but ability to use at all times and in all places. *Light weight* is a feature desired by every Mining Engineer, provided it can be combined with *accuracy* and *durability*.

This is accomplished in the Gurley Light Mountain Transits—especially popular with Mining Engineers everywhere.

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Investigate This New Mining Transit



It is not too heavy to handle easily nor too light to be rigid. The NEW.

Intermediate H. & B. Transit

"For Mining Engineers"

is specially designed to meet their every requirement.

Has all the distinctive advantages of the improved H. & B. Transit. Graduations upon inlaid, solid sheet silver, dust and rain guards to leveling screws and telescope objective slide, detachable leveling head.

Backed by nearly half a century successful manufacturing experience.

Guarantee your next instrument investment by investigating this new H. & B. now. A postal request brings complete details and actual photograph. No obligation.

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Established 1870. Philadelphia, a.

A DIGGER AT LOW COST



A small amount of money invested in our Derrick Excavator means practically a steam shovel or dipper dredge at your disposal. Our outfit is guaranteed to accomplish the same results as the more expensive equipment. Write for our catalog and learn more about our outfit.

UNION IRON WORKS

NEWARK STREET HOBOKEN, N. J.

INDUSTRIAL SECTION

Pulverized Fuel Combustion

The apparatus shown in the illustration is a complete equipment for supplying coal to boiler and metallurgical furnaces, cement and lime kilns, dryers, and to all furnaces requiring a temperature and quality of flame under prompt and easy control.

It is claimed that the Aero Pulverizer makes practicable the highest efficiency obtainable from burning coal. It makes coal burn like a gas, with a flame, the physical and chemical character of which is regulable—a flame that may be elongated or shortened, thus placing the zone of highest temperature where needed—a flame that may be made oxidizing, reducing, or neutral, as occasion may require.

The coal is burned as pulverized, and there is no storage of the powder with its attendant hazard. Artificial drying before pulverizing is not



necessary if the coal supply be sheltered from rain and snow. Where the Aero is used, it is wholly a furnace question whether a dryer should be installed; it is not at all a pulverizing or storage question.

Labor is reduced to a minimum.

Slack coal at low cost yields its last B.T.U.

The Aero Pulverizer approaches the subject of coal burning from the theoretical side, and, therefore, pulverizes the coal to an impalpable powder, and surrounds each of its minute particles with the amount of air which will furnish just the required oxygen. The fineness of the pulverization may be regulated by attention to the dampers which control the movement axially of the air within the machine. If that movement is slow, the centrifugal force keeps all coarse particles at the periphery, but if the movement axially is rapid, it in part overcomes centrifugal force and draws through the machine a coarser grade of material. Powdered coal and air in regulable proportions are intimately mixed in the pulverizer, and the mixture reaches the furnace instantly it leaves the pulverizer. Thus, the Aero system is emancipated from not only the dryer, but the

ADVANTAGES TO MEMBERS OF OUR ADVERTISING SECTION

It is our intention to make the advertising section of the *Bulletin* as comprehensive as possible, so that members can turn to it for complete information as to where to buy the best mining and metallurgical supplies at the lowest prices. No firms or supplies of dubious reputation are advertised in the *Bulletin*. If members, in buying any material, will refer to the advertisement of that material in the *Bulletin*, or if they do not find it advertised in the *Bulletin*, will, when ordering, ask the manufacturers why the goods were not advertised in the Institute publication, it will be of material benefit to all of us, because the chief reason why advertisers are unable to trace direct results is because purchasers do not mention the *Bulletin* when writing. Finally, if members, when in need of anything whatsoever, will write to the Institute headquarters, an effort will be made to send them a complete list of manufacturers able to supply the desired materials of good quality. The furnishing of these lists will be a benefit to our Advertising Department, as well as to the members. We need scarcely call attention to the advantages obtained by the members from the money which advertising brings in, because all this money is returned directly to the members in one form or another.



LEYNER DRILL SHARPENERS

The Leyner Sharpener is an economical necessity in mining operations. It sharpens and makes all kinds of bits, true to gauge and form; and does the work more rapidly and cheaply than can be done by hand. It also shanks the steel.

Properly sharpened steel facilitates more rapid drilling and relieves the drilling machines of unnecessary wear and tear.

The Leyner Sharpener has no real competition. It is built in two sizes.

Bulletin No. 4022

INGERSOLL-RAND COMPANY

NEW YORK

LONDON

24-LDS

BULLETIN, A. I. M. E.—INDUSTRIAL SECTION

powdered coal conveyor apparatus, the storage bin, the mixing chamber and the feeding apparatus with the power units required for the several operations, which are incident to all central station pulverizing systems. The manufacturers state that there is no smoke, no carbon in the ash, no CO in the flue gases, and only a trace of O, no appreciable excess air is admitted to reduce the temperature of the products of combustion. There is no opening of doors, no intermittent firing, no banked fires, no delay in meeting a sudden overload.

The efficiency of heat from combustion is directly as the rapidity of combustion. The decaying log is a form of combustion so slow that its efficiency is not noticeable. The greatest rapidity and efficiency is obtained by bringing each atom of carbon in contact with two atoms of oxygen, and no more, under conditions permitting chemical union, and the conditions produced by the Aero measurably approach the theoretical in this respect.

The Aero Pulverizer consists of three interiorly communicating chambers (Type "E" has four) of successively increasing diameters, in which revolve paddles on arms of correspondingly increasing lengths. The separate chambers are in fact separate pulverizers on a single shaft, each succeeding pulverizer having greater speed at its periphery and therefore greater power for fine grinding. An additional chamber contains a fan, the function of which is to draw the more finely pulverized material successively from one chamber to the next, and finally, to deliver it through a pipe connection to the furnace under the impetus of a forced draft. The separate pulverizers and fan are inclosed in one steel cylinder. A regulable feed mechanism accurately controls and varies the quantity of coal admitted to and delivered by the machine. The feed mechanism is exact and uniform in its operation and is easily adjusted to meet even minute variations in the fuel requirement. Two regulable inlets in the feed mechanism admit the air required for fine grinding. An auxiliary inlet between the last work chamber and the fan, controlled by a damper, admits such additional air as is required for combustion. The air dampers with the feed give perfect regulation of the flame within a wide range.

The discharge is either right or left hand, according to specification.

The Aero Pulverizer is dust-proof, is strongly built, and is arranged for easy repair to the parts susceptible to wear, the cost of which is small. The bearings are generous and self-oiling. The drive may be by belt or by direct connected motor.

The pulverizer may be located either in front of the furnace or at either side, or above, or below.

The connection between the pulverizer and the furnace is usually a galvanized iron pipe from the pulverizer discharge to the furnace. No additional feeding or mixing apparatus could serve any useful purpose, for the powdered coal and air are intimately mixed in the pulverizer. The furnace end of the discharge pipe is made of such size and shape as the furnace condition requires.



ALBANY GREASE

For the lubrication of blower fans, engines, pumps, hoists, drills, etc. It gives the highest lubricating service at an exceptionally low cost. You can experiment at our expense. Simply ask for a quantity of Albany Grease and an Albany Cup. They'll be sent without charge. Write now.

**ALBANY LUBRICATING CO.
NEW YORK**



The Best and Most Innocent are Often Wrongfully Accused.

For instance, take a case where a belt was allowed to run slack enough to slip causing "pulley burn." There was no actual evidence that this had happened after the one responsible tightened her up—his "boss" was a man who knew nothing of belts and was easily convinced that the concern who sold the belting "put one over."

Schieren's Duxbak Waterproof Leather Belting is less liable to pulley burn because it keeps a uniform length in all kinds of weather after once being properly fitted to the pulleys.

Try a Duxbak on your Centrifugal Pumps, Jaw Crushers, Gyrotory Breakers, Rolls, Chilean Mills, Conical Mills, Tube Mills, Stamp Mills, Trommels, Concentrating Tables, Jigs, Vanners, Blowers, Blowing Engines, Air Compressors, Gas and Steam Engines, Motors, Dynamos and you'll appreciate its value as a help to production.

Charles A. Schieren Co., 30-38 Ferry St., New York City

IMPORTANT ANNOUNCEMENT!

A group of Weston Portable Electrodynamometer Instruments of Precision, comprising:

Model 370 A. C. and D. C. Ammeter, Write for Our Bulletin No. 2003

Model 329 Polyphase Wattmeter, Write for Our Bulletin No. 2002

Model 341 A. C. and D. C. Voltmeter, Write for Our Bulletin No. 2004

Model 310 Single Phase and Direct Current Wattmeter, Write for Our Bulletin No. 2002

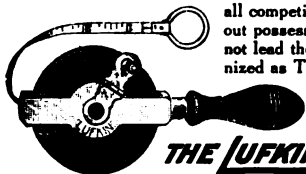
This group warrants the careful investigation of any one interested in the Art of Precise Electrical Measurement.

Weston Electrical Instrument Co., 37 Weston Ave., Newark, N. J.

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LUFKIN

TAPES



could not have led the field in spite of all competition for a quarter of a century without possessing sterling quality. They could not lead the field to-day if they were not recognized as **THE BEST—ABSOLUTELY**

INSTANTANEOUS READINGS. Metal Lined Cases. ACCURATE, DURABLE, DEPENDABLE.

On Sale Everywhere.

Catalog on Request.

THE LUFKIN RULE CO.

SAGINAW, MICH.

New York; London, Eng.; Windsor, Can.

Each Aero Pulverizer is designed as a complete powdered fuel unit for one furnace. They are built in four standard sizes which thus far have met varying furnace requirements. Other sizes will be furnished, if desired.

Conditions are sometimes adverse, requiring the use of very wet coal, and at the same time a heavy overload, and therefore motors of higher power than are needed for normal conditions are recommended by the manufacturers, the Aero Pulverizer Co., 63 Broadway, New York.

Slide Rule and Slope Chart

The illustrations show the Chemist's Duplex Slide Rule, designed by Dr. R. Harman Ashley, and the Brunton Slope Chart.

The rule represents the successful completion of a long series of experiments to adapt the logarithmic and cologarithmic scales to the rapid solution of the problems encountered by the chemist.

Through the adaptation of the K. and E. Duplex Slide Rule it is claimed that the solution of the commoner problems has been greatly simplified, while the scope of the rule has been increased by the inclusion of a large number of chemical symbols. The symbols on the rule, being arranged in the order of the molecular weight of the elements, are easily found, while the application of a very simple rule enables the chemist to locate other symbols of less frequent occurrence. Aside from the solution of the chemical problems above referred to, any arithmetical problems solvable by logarithms are readily and accurately done with a minimum number of settings.



The rule carries 138 chemical symbols, which include the common acids, bases, salts, oxides, and elements employed by the average chemist. As each symbol has its individual position corresponding to the logarithm of the molecular weight, the number of permutations and combinations possible covers the requirements of almost any problem.

By using the logarithmic and cologarithmic scales in conjunction with the chemical gauge points, problems in stoichiometry, such as gravimetric analysis, volumetric analysis, equivalents, percentage composition, conversion factors, volume of gas from a given weight of substance at different temperatures and pressures, and many other analogous problems are readily solved.

The Chemist's Duplex Slide Rule is accompanied by a manual giving the theory of its use together with numerous examples of both arithmetical and chemical nature. The rule is designed to work as simply as possible and is easily mastered. It is engine divided, and may be relied upon to an extent equal to the operations of ordinary analysis. For this

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Zinc Ores, Carbonates, Sulphides and Mixed Ores, Copper Ores, Copper Matte, Copper Bullion, Lead Bullion, Lead Ores, Antimony Ores, Iron and Manganese Ores, Copper, Spelter, Antimony, Antimonial Lead, Sulphate of Copper, Arsenic, Zinc Dust.

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DORR CLASSIFIERS THICKENERS AGITATORS

In use by Institute Members all over the World

The Dorr Cyanide Machinery Co.

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Cordeau- Bickford Shot

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Instantaneous Detonat-
ing Safety Fuse

FOR
SAFETY and
EFFICIENCY

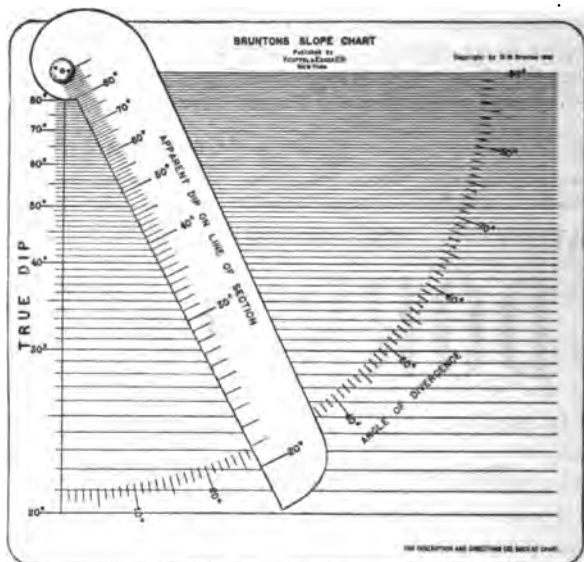
THE ENSIGN BICKFORD CO., Simsbury, Conn.

Original Manufacturers of Safety Fuse

Established 1836

reason it may be employed in calculating results or in checking calculations made by other methods.

The Brunton Slope Chart is for the determination of the apparent angle of sections when cut at planes of varying degrees of divergence from the dip.



The Chart enables the user instantly to obtain the apparent dip from the true dip, or vice versa; mechanically solving the equation:

$$\tan C^\circ = \cos A^\circ \tan B^\circ,$$

in which C is the apparent dip; A , the angle of divergence; and B the true dip.

In addition to its use in the preparation of maps and geological sections, the chart is also extremely useful for giving the valley angles in hoppers, ore bins, etc.

Additional information may be obtained from the manufacturers, the Keuffel & Esser Co., Hoboken, N. J.

Vertical Centrifugal Pump

In the June, 1915, Bulletin (No. 154) of the A. S. Cameron Steam Pump Works, New York, is given a description of their double suction vertical centrifugal pump, herewith illustrated.

The vertical arrangement of this pump, with the actual pump below the water level, is recommended where automatic operation is demanded, as the pump is always primed and ready for instant service.

From the Bulletin we learn that the pump casing is volute in form, divided vertically, with suction and discharge nozzles cast integral with the back half. The front half may be quickly removed without disturbing the bearings or pipe connections. Both halves of the casing are provided with heavy flanges carefully faced and drilled, bolted and doweled

Barsteel Wins



Replacing a brake shoe on a Barsteel Locomotive

“We want your Barsteel Locomotive

because it has the open frame so that we can replace your M. C. B. brake shoe in half an hour or less. We can't afford to take any chance of being held up any longer than that when the coal is coming. We like the way everything about your locomotive is made accessible.” This statement was made to us by a West Virginia coal operator when he duplicated an order for several 15-ton Barsteel Locomotives.

Barsteel locomotives make friends everywhere, they stand up to the work for a long time and when a repair does become necessary it is easy to make it.

For information on Barsteel Locomotives write either Company:

The Baldwin Locomotive Works

Philadelphia, Pa.

Westinghouse Electric & Mfg. Co.

East Pittsburgh, Pa.

Barsteel—The Frame That Makes Inspection Easy.

BULLETIN, A. I. M. E.—INDUSTRIAL SECTION

together. All passages through the pump are designed to give the highest possible efficiency.

The impeller is of the double suction inclosed type, specially designed for the conditions of service. It is cast in one piece, accurately machined and polished all over to insure perfect rotative balance, and the water passages are carefully smoothed to reduce friction. It is secured on the shaft by a feather key and firmly held in place by bronze nuts formed on the shaft protecting sleeves.



The shaft is made of hammered open-hearth steel of ample size to easily transmit the maximum power required. It is accurately machined and ground, and provided with removable bronze sleeves where it passes through the stuffing boxes. These bronze sleeves are screwed on the shaft against the impeller.

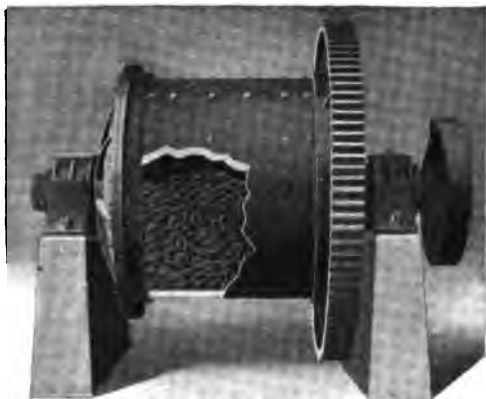
The stuffing box is of ample depth and provided with water seal, consisting of a lantern gland connected to the water from the discharge side of the pump.

The stuffing box gland is fitted with swing bolts to give quick and easy access to the stuffing box.

The pump is fitted with two bearings, one on each side of the casing. The bearing bodies are equipped with removable caps, and the upper bushing is split and lined with high-grade bearing metal. The lower bearing will contain a heavy lignum vitæ sleeve. The bearings will be of ample proportions to prevent heating.

The bearings are separate from the casing, and located sufficiently distant from the stuffing boxes to permit of easy adjustment of the glands.

A copy of Bulletin No. 154, giving additional information, may be obtained by addressing the A. S. Cameron Steam Pump Works, 11 Broadway, New York.



7 feet × 6 feet Traylor Short Type Mill.

TRAYLOR TUBE MILLS GRINDS WITHOUT SLIMES!

100% Crushing Efficiency
Throughout the Entire
Length of the Mill.
No Drag on the Liners.
No Waste of Power.
No Slime Production.

Traylor Improved Short Tube Mills have been proven by actual practise to be especially suited for medium fine grinding, with large capacity and small consumption of pebbles.

Traylor Short Tube Mills embody many important features and are built in sizes ranging from 4 feet to 7 feet in diameter and 6 feet long.

Send us your requirements and let us do the rest.

TRAYLOR ENGINEERING & MANUFACTURING CO.

MAIN OFFICE & WORKS—ALLEETOWN, PA.

N. Y. Office: 28 Church Street

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Sullivan Underground Compressors and Sullivan Hammer Drills



Sullivan Portable Electric-Driven Compressor "WK-2"

hose to a Sullivan "ROTATOR," or a Sullivan stoper, start your motor, and you'll be ready to shoot in no time.

form a wonderfully efficient combination for light development work in the electrically-equipped mine.

If you have a winze to sink or a raise to drive, or any other light rock drilling to do far from your main air line, just hook your "WK-2" Compressor to the feed wire, couple your



Details in Bulletin 858-P

SULLIVAN MACHINERY CO.

122 SO. MICHIGAN AVE.

CHICAGO

A Goodrich Belt Record

The problem of catching and carrying units weighing 125 lb. dropping from a height of 4 ft. at the rate of 3,200 a day faced the California Hawaiian Sugar Refining Co.; nearly ten years ago.

On Feb. 15, 1906, this company installed a 36-in. 7-ply Goodrich Conveyor Belt and since that time it has caught and carried 2,000,000,000 lb. of sugar (every bit of it having fallen that 4 ft.) before there were any radical signs of wear. It was a big belt and was quite an investment, but the service it gave exceeded expectations.

About one month ago this same concern entered an order with The B. F. Goodrich Co. for another *long-lived* belt. The man in the accompanying picture is 5 ft. 11½ in. tall, which gives one an idea of the size of the roll. A belt 1,443 ft. long, 36 in. wide and weighing 11,983 lb. net is a huge proposition.

Figuring 8 hr. run without any delays, a bag of sugar would drop on to the belt every 9 sec., and if the speed of the belt were such that it could



make 26 or 27 revolutions every 8 hr., the bags would rest at intervals of 12 ft. apart.

At the given speed, therefore, the belt will carry a continuous load of 60 bags or 7,500 lb., and every 9 sec. its heavily burdened surface will receive the sudden jolt and strain caused by 125 lb. dropping 4 ft. Such service is unusual in its severity and is evidence of the strength the B. F. Goodrich Co. have succeeded in building into their conveyor belting.



JEFFREY Stepped Multi-Bladed Mine Fans

**Save 10% to 30%
in Power Bills.**

With their stepped series of blades, they receive and accelerate the air without shock and discharge it at low velocity, which means maximum economy of operation.

Figuring the cost of operating a 100 H.P. Fan at \$5000.00 yearly, a saving of 20% in power means a saving of \$1000.00 at the end of the year.

Send for Bulletin No. 110-2, giving full particulars.

JEFFREY MFG. CO., 902 N. Fourth St., Columbus, O.

New York Boston Philadelphia Pittsburgh Chicago Birmingham Denver Montreal



IN REGARD TO WIRE ROPE

Most engineers are aware that service clearly reveals the quality of material and precision with which such rope is laid up.

ROEBLING

wire ropes are high grade special ropes that are the products of over

**76 YEARS DEVOTED TO
ENGINEERING STUDY.**

For Derricks, Cranes, Hoists, Elevators, Cableways, Tramways, Power Transmission, Etc.,

Roebling Rope Gives Highest Service.

WRITE FOR PRICES

JOHN A. ROEBLING'S SONS CO.
TRENTON, N. J.

A New Catalogue

Ordinarily, the issue of a new catalogue in the engineering field causes only a ripple of interest on the part of those whose every day work brings them in contact with the product of the manufacturers. Yet the issuance of a new and up-to-the-minute catalogue of engineering instruments is surely a visible sign of progressiveness, not only in maintaining the high standard of the appliances, but in giving the engineering profession the means of making an intelligent comparison when purchasing surveying instruments.

The new catalogue just published by Heller & Brightly, Philadelphia, Pa., contains 125 pages of well illustrated description of H. & B. instruments, together with information of value to the experienced engineer as well as the beginner.

A copy of the catalogue will be forwarded on request.

Trade Catalogues

Rock Drills and Accessories.—Denver Rock Drill Mfg. Co., Denver, Colo. Several well-written and illustrated bulletins on their products, including "Clipper" drills, Clark Air Meters, Waugh Valveless Stopers. Pluggers, and Drifters. Valuable data of interest to the mine operator are contained in these bulletins.

"*Continuous Counter-Current Decantation*" is the title of a 38-p. catalogue published by the Dorr Cyanide Machinery Co. of Denver, Colo. Data of interest to the mining man on the Dorr Classifier, Thickener, and Agitator, are included. Illustrated.

Electric Mining Machinery.—The Goodman Mfg. Company, Chicago, Ill. Six bulletins, which in reality are catalogues, well written and profusely illustrated, descriptive of Single-Motor Locomotives, 32 pp.; Two-Motor Locomotives, 48 pp.; Gathering Locomotives, 52 pp.; Rack Rail Haulage, 52 pp.; Straight-face Machine, 25 pp.; Shortwall Machines, 32 pp.; and Chain Breast Machines. A complete set will be sent to all mining men addressing the company.

The Jeffrey Manufacturing Co., Columbus, Ohio, manufacturers of tippie and coal-mine machinery, have issued Bulletin No. 110 on the subject of Mine Ventilation, calling particular attention to their new Stepped Multi-Bladed Wheel Type Fan, which, by means of its stepped series of blades, receives and accelerates the air without shock and discharges it at low velocity, insuring maximum economy of operation. This bulletin also illustrates and describes their line of Centrifugal Booster Fans, and contains valuable data compiled to give the mine superintendent, manager, or those interested in this line of work a comprehensive idea of the results obtained from various sizes of Jeffrey Mine Fans. A free copy of this bulletin may be obtained by addressing the home office of the company at 902 N. Fourth St., Columbus, Ohio, or any of their branch offices.

The Slogan of the Cameron—"Character: The Grandest Thing"

ECONOMICAL AND DEPENDABLE PUMPS



Down in the mine where the service is continuous and exacting, you need pumps that are not only economical, but absolutely dependable.

Cameron Centrifugals

have just those features of design and construction which make them superior for this class of work.

Note the simplicity of the Multi-Stage Turbine type, illustrated. The horizontally split casing allows quick, easy access to all working parts. Unusual care in choosing material and thoroughness of manufacture insure low upkeep costs.

Get your pumping done at the least expense—install the Cameron.

Ask for full particulars and Bulletin No. 151.

A. S. CAMERON STEAM PUMP WORKS

11 BROADWAY, NEW YORK

10

Offices the World Over

Maintenance

¶ It can be made almost negligible.

¶ Or it can be made excessive.

¶ Sometimes it's the fault of the hoist.

¶ But isn't it often the fault of *contributing* conditions, or the operator.

¶ Take VULCAN hoists, for instance. We take the utmost pains in the making. We test and analyse every casting, every forging; our own specifications are more rigid than yours. There's not a thing left undone to make maintenance *low*.

¶ And if it is not low, it's generally traceable to outside conditions that can be remedied easily.

¶ Keep your hoist clean, well-oiled and greased, and test all the nuts and bolts now and then, and give the bearing wedges a look.

¶ Then you'll see maintenance worries disappear.

VULCAN IRON WORKS
1744 Main Street
Wilkes-Barre, Pa.

A DIRECTORY OF MINING AND

ALBANY GREASE	ALBANY LUBRICATING CO. Adam Cook's Sons, Proprietors 708-10 Washington Street New York City Manufacturers of Albany Grease, Cook's Lubricant and Albany Automobile and Machinery Oils. Manufacturers and refiners of lubricants for every condition and service. Write for descriptive matter.
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MINING MACHINERY POWER AND ELECTRIC MACHINERY	ALLIS-CHALMERS MANUFACTURING CO. Milwaukee, Wisconsin. MINING MACHINERY of Every Type. Complete Power and Electrical Equipments. For all Canadian business refer to Canadian Allis-Chalmers, Limited, Toronto, Ont.
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PUMPS	A. S. CAMERON STEAM PUMP WORKS 11 Broadway, New York. CAMERON VERTICAL PLUNGER SINKING PUMPS, for shaft sinking. CAMERON HORIZONTAL PLUNGER STATION PUMPS, for handling gritty water.
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MOTORS FOR EVERY PURPOSE	CROCKER-WHEELER CO. Ampere, N. J. Manufacturers of Electric Motors, Generators and Transformers. Alternating and Direct Current.
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ASSAYERS AND CHEMISTS SUPPLIES	THE DENVER FIRE CLAY CO. Denver, Colo. Salt Lake City, Utah. Manufacturers of ASSAY SUPPLIES, CRUCIBLES, SCORIFIERS, Muffles, Fire Brick, Scientific Apparatus, Chemical Apparatus, Heavy Chemicals, C. P. Chemicals, Glass-blowing, etc. Write to-day for Catalogue.
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ROCK DRILLS DRILL SHARPENERS AIR METERS STEEL HOSE ETC.	DENVER ROCK DRILL MANUFACTURING CO. Denver, Colo. El Paso, Tex. New York City. Salt Lake City, Utah. San Francisco, Cal. MANUFACTURERS OF WAUGH DRILLS.
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METALLURGICAL EQUIPMENT

THE DORR CYANIDE MACHINERY CO.

New York DENVER, COLO. London
Machinery in use for Cyaniding. Wet gravity Concentration, Flotation, Leaching Copper ores and many non-metallurgical industrial processes.

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THICKENERS
AGITATORS**

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Manufacturers of the EDISON STORAGE BATTERY for Mine Haulage. Write for descriptive bulletin.

**EDISON
STORAGE
BATTERY**

THE ENSIGN BICKFORD CO.

Simsbury, Conn.
Manufacturers of
SAFETY FUSE CORDEAU-BICKFORD
For all Blasting Operations
Fuse for special requirements

**CORDEAU-
BICKFORD
—
SAFETY FUSE
FOR
ALL BLASTING
—
BLASTING
SUPPLIES**

GENERAL ELECTRIC CO.

Schenectady, N. Y.
**ELECTRIC MINE LOCOMOTIVES. ELECTRIC
MOTORS for Operating Mining Machinery.**

**ELECTRIC
MINE
LOCOMOTIVES**

GOODMAN MANUFACTURING CO.

Chicago, Illinois.
**ELECTRIC AND AIR POWER COAL CUTTERS.
ELECTRIC MINE LOCOMOTIVES.
POWER PLANTS.**

**ELECTRIC
COAL CUTTERS

MINE
LOCOMOTIVES**

THE B. F. GOODRICH CO.

Akron, Ohio.
Goodrich "Longlife" "Mazecon" & "Grainbelt" CONVEYOR BELTS will handle more tons per dollar of cost than any other belts made.

**CONVEYOR
BELTS**

A DIRECTORY OF MINING AND

SURVEYING INSTRUMENTS FOR MINING ENGINEERS	W. & L. E. GURLEY Troy, N. Y. Branch, Seattle, Wash. Manufacturers of "Gurley Quality" transits and levels. Complete catalogue covering mining instruments sent upon request. Have one on file.
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BRICK FIRE CLAY SILICA MAGNESIA CHROME	HARBISON-WALKER REFRACTORIES CO. Pittsburgh, Penna. Refractories of highest grade for Blast Furnace and the Open Hearth, Electrical Furnaces, Copper Smelting plants, Lead Refineries, Nickel Smelters, Silver Slimes and Dross Furnaces, Alloy Furnaces, as well as all other types in use in the various metallurgical processes.
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INSTRUMENTS FOR MINING ENGINEERS	HELLER & BRIGHTLY Philadelphia, Pa. INSTRUMENT MAKERS Mathematical, Optical, Engineering, Surveying and Astronomical Instruments.
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SPELTER SHEET ZINC SULPHURIC ACID	ILLINOIS ZINC CO. Peru, Ill. Manufacturers of SPELTER, SHEET ZINC and SULPHURIC ACID.
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COAL MINING MACHINERY	INGERSOLL-RAND CO. 11 Broadway, New York. "Return-Air" Pumps, Coal Shearers, Pneumatic Hoists, "Electric-Air" Drills, Coal Punchers, Pneumatic Tools, "Calyx" Core Drills, Plug Drills, Hammer Drills, Tamp- ing Machines, Rock Drills, Air Lift Pumps.
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COAL MINING MACHINERY	THE JEFFREY MFG. CO. Columbus, Ohio. Electric and Air Power Coal Cutters and Drills, Car Hauls, Coal Tipples, Coal Washeries, Larries, Screens, Cages, Crushers, Elevators, Conveyors, Fans, Hoists, Pumps, etc.
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METALLURGICAL EQUIPMENT

LEAD LINED IRON PIPE CO.

Wakefield, Mass.

LEAD LINED IRON PIPE, LEAD LINED IRON VALVES—for Acids and Corrosive Waters.

LEAD LINED
IRON PIPE
AND VALVES

A. LESCHEN & SONS ROPE COMPANY

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ALPHABETICAL LIST OF ADVERTISERS

	PAGE
Albany Lubricating Co.	5
Allis-Chalmers Manufacturing Co.	16
Beer, Sondheimer & Co.	7
A. S. Cameron Steam Pump Works	15
Crocker-Wheeler Co.	16
Denver Fire Clay Co.	16
Denver Rock Drill Co.	16
Dorr Cyanide Machinery Co.	7
Du Pont Powder Co.	3
Edison Storage Battery Co.	Inside Back Cover
Ensign Bickford Co.	7
General Electric Co.	Outside Back Cover
Goodman Manufacturing Co.	17
B. F. Goodrich Co.	Inside Front Cover
W. & L. E. Gurley	1
Harbison-Walker Refractories Co.	18
Heller & Brightly	1
Illinois Zinc Co.	18
Ingersoll-Rand Co.	3
Jeffrey Manufacturing Co.	13
Lead Lined Iron Pipe Co.	19
A. Leschen & Sons Rope Co.	19
Lufkin Rule Co.	5
Mashek Engineering Co.	19
Robins Conveying Belt Co.	19
John A. Roebling's Sons Co.	13
P. H. & F. M. Roots Co.	20
E. G. Ruehle & Co.	7
Chas. A. Schieren.	5
Sullivan Machinery Co.	11
Terry & Tench Co.	20
Traylor Engineering & Manufacturing Co.	11
Union Iron Works	1
L. Vogelstein & Co.	23
Vulcan Iron Works.	15
Westinghouse Electric & Mfg. Co.	9
Weston Electrical Instrument Co.	5



DECEMBER, 1915



Bulletin of the American Institute of Mining Engineers

PUBLISHED MONTHLY

SECTION I

	PAGE		PAGE
New York Meeting.....	iii	Employment.....	xxii
Proceedings of San Francisco Meeting	iv	Local Section News.....	xxiv
Naval Consulting Board.....	xiii	Affiliated Student Society Notes.....	xxiv
The Engineering Foundation.....	xv	Library.....	xxvi
International Engineering Congress.....	xv	Membership.....	xxix
2nd Pan-American Scientific Congress.....	xvii	Institute Committees.....	xlvi
Personal.....	xxi	Officers and Directors.....	liv

SECTION II.—PAPERS AND DISCUSSIONS

Protecting California Oil Fields from Damages by Infiltrating Water. By R. P. McLaughlin	2313
Notes on Flotation. By J. M. Callow	2321
The Effect of Carbon on the Physical Properties of Heat-Treated Carbon Steel. By J. H. Nead	2341
The Determination of Grain Size in Metals. By Zay Jeffries, A. H. Kline, and E. B. Zimmer	2359
Magnetic Studies of Mechanical Deformation in Certain Ferromag- netic Metals and Alloys. By H. Hanemann and Paul D. Merica	2371
Washed Metals. By Henry D. Hibbard and Edward L. Ford	2387
Manganese-Steel Castings in the Mining Industry. By Walter S. McKee	2399
Discussions of Papers of the San Francisco Meeting (See first page of text).	
Industrial Section Interpolated in Advertising Section.	

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DISCUSSION OF PAPERS PRESENTED AT THE SAN FRANCISCO MEETING SEPTEMBER, 1915

MCLAUGHLIN, R. P.—Protecting California Oil Fields from Damage by Infiltrating Water. Discussed by M. E. Lombardi, A. F. L. Bell, A. C. McLaughlin, R. P. McLaughlin, William A. Williams, Mark L. Requa, C. D. Keen	2413
BEST, W. N.—Petroleum as Fuel Under Boilers and in Furnaces. Discussed by William A. Williams, Hennen Jennings, David T. Day, Mark L. Requa, A. F. L. Bell, W. N. Best	2420
JOHNSON, ROSWELL H.—Sliding Royalties for Oil and Gas Wells. Discussed by William A. Williams, A. C. McLaughlin, A. F. L. Bell, Roswell H. Johnson	2423
BELL, ARTHUR F. L.—Important Topping Plants of California. Discussed by William A. Williams, David T. Day, A. F. L. Bell	2426
LOMBARDI, M. E.—The Cost of Maintaining Production in California Oil Fields. Discussed by C. D. Keen, Mark L. Requa, H. M. Bacon, M. E. Lombardi	2427
HUNTLEY, L. G.—Oil, Gas, and Water Contents of Dakota Sand in Canada and United States. Discussed by E. W. Shaw	2428
WEAVER, CHARLES E.—The Possible Occurrence of Oil and Gas Wells in Washington. Discussed by Milnor Roberts, A. C. McLaughlin, W. N. Best, David T. Day, J. B. Tyrrell	2431
DUMBLE, E. T.—The Occurrences of Petroleum in Eastern Mexico as Contrasted with Those in Texas and Louisiana. Discussed by E. L. DeGolyer	2434
BOSQUI, F. L.—Metallurgical Practice in the Witwatersrand District, South Africa. Discussed by Sidney J. Jennings, C. W. Merrill, Hennen Jennings, W. A. Caldecott, A. J. Clark, G. H. Stanley, W. R. Dowling	2436
CLARK, ALLAN J.—Notes on Homestake Metallurgy. Discussed by G. H. Clevenger, John A. Fulton, A. J. Clark	2438
SHARWOOD, W. J.—A Rule for Governing Cupellation Losses. Discussed by Frederic P. Dewey	2454
THORNHILL, E. B.—Recovery of Mercury from Amalgamation Tailings, Buffalo Mines, Cobalt. Discussed by D. B. Huntley, H. G. S. Anderson	2455
LASS, W. P.—Electric Furnace for Gold Refining at the Alaska-Treadwell Cyanide Plant. Discussed by Robert M. Keeney, R. S. Wile	2456
RICKETTS, L. D.—Some Problems in Copper Leaching. Discussed by Frederick Laist, T. T. Read, E. H. Hamilton	2459
ADDICKS, LAWRENCE—Roasting and Leaching Concentrator Slimes Tailings. Discussed by L. D. Ricketts, E. H. Hamilton, Frederick Laist, Charles Butters, T. T. Read, Lawrence Addicks	2460
THOMPSON, A. PERRY.—The Occurrence of Covellite at Butte, Mont. Discussed by Arthur A. Bakke, H. W. Turner, C. F. Tolman, A. C. Lawson, A. F. Rogers, L. C. Graton, J. C. Ray, A. Perry Thompson	2464
DONNELLY, THOMAS F.—The Copper Deposits of San Cristobal, Santo Domingo. Discussed by F. Lynwood Garrison	2473
DAKE, C. L.—The Formation and Distribution of Bog and Iron-Ore Deposits. Discussed by A. C. Lawson, G. H. Cox, C. L. Dake	2475
SOMERS, R. E.—Geology of the Burro Mountains Copper District, New Mexico. Discussed by A. C. Lawson, L. C. Graton	2476
MITER, CHARLES A.—Ventilation of the Copper Queen Mines. Discussed by Gerald Sherman, Joseph P. Hodgson	2477
BLACKNER, LESTER A.—Underground Mining Systems of Ray Consolidated Copper Co. Discussed by Sidney J. Jennings	2478
BRETHERTON, S. E.—The Advantages of High-Lime Slags in the Smelting of Lead Ores. Discussed by Anton Eilers	2479
MITINSKY, A. N.—Some Suggestions Regarding the Determination of the Properties of Steel. Discussed by Lawford H. Fry, H. V. Wille, Henry M. Howe, Mansfield Merriman, James E. Howard, George K. Burgess, Gaetano Lanza	2481
JONES, CHARLES COLCOCK.—The Pacific Coast Iron Situation. Discussed by D. A. Lyon, Carl H. Booth, Bradley Stoughton, H. W. Lash, J. W. Richards, Mr. Malm, Mr. Randall, Mr. Willeston, Charles Colcock Jones	2498
LYON, D. A. AND KEENEY, R. M.—Electrometallurgical Industries as Possible Consumers of Electric Power. Discussed by Lawrence Addicks, Carl H. Booth, J. W. Beckman, Kenneth Seaver, J. W. Richards, D. A. Lyon	2502
KRANZ, WILLIAM G.—The Electric Furnace in the Foundry. Discussed by M. Petinot, Carl H. Booth, J. W. Richards, H. W. Lash	2507
GATHMANN, EMIL.—The Commercial Production of Sound, Homogeneous Steel Ingots and Blooms. Discussed by H. W. Lash, J. W. Richards	2511
TIEMANN, HUGH P.—Conversion Scale for Centigrade and Fahrenheit Temperatures. Discussed by J. W. Richards, Hugh P. Tiemann	2512
BOECK, P. A.—The Thermal Insulation of High-Temperature Equipment. Discussed by Lawrence Addicks, J. W. Richards, A. H. Krieger, J. W. Beckman, Henry Kreisinger and J. F. Barkley, William A. Williams, Kenneth Seaver	2513
SEAYER, KENNETH.—Manufacture and Tests of Silica Brick for the Byproduct Coke Oven. Discussed by J. W. Richards, William A. Williams, Kenneth Seaver	2519



MEMBERS OF THE INSTITUTE AND GUESTS AT CEREMONIES IN COURT OF ABUNDANCE, PANAMA-PACIFIC INTERNATIONAL EXPOSITION,
SAN FRANCISCO, SEPT. 17, 1915.

BULLETIN OF THE AMERICAN INSTITUTE OF MINING ENGINEERS

PUBLISHED MONTHLY

No. 108

DECEMBER

1915

Published Monthly by the American Institute of Mining Engineers at 212-218 York St., York, Pa., H. A. WISOTSKY, Publication Manager. Editorial Office, 29 West 39th St., New York, N. Y., BRADLEY STOUTERON, Editor. Cable address, "Aime," Western Union Telegraph Code. Subscription (including postage), \$10 per annum; to members of the Institute, public libraries, educational institutions and technical societies, \$5 per annum. Single copies (including postage), \$1 each; to members of the Institute, public libraries, etc., 50 cents each.

Entered as Second Class matter January 28, 1914, at the Post Office at York, Pennsylvania, under the Act of March 3, 1879.

NEW YORK MEETING, FEB. 14 TO 17, 1916

The 112th Meeting of the Institute, and the Annual Business Meeting, will be held at the Headquarters of the Institute in New York City, Feb. 14 to 17, 1916.

The Chairmen of several of the technical committees report that papers on subjects of timely interest in their respective branches have been promised. David H. Browne, who was appointed Chairman of the Committee on Arrangements, has organized his committee for active work.

Members are urged to make their plans now to be present at this meeting to avail themselves of the pleasant opportunity for greeting old friends and making new ones among those engaged in the same profession.

1916 DUES

In accordance with the provisions of the Constitution, notice is hereby given to all members that the dues of the year 1916 will be payable on Jan. 1, 1916, at the office of the Secretary. The dues are \$10, and the sum of \$3 additional is asked of those members who desire to have the three volumes, which will be issued in 1916, bound in the Institute's standard half-morocco binding.

Volume LI of the Transactions is now in the press and will be sent in January to all members whose dues for the year 1916 are paid at that time.

PROCEEDINGS OF THE ONE HUNDRED AND ELEVENTH MEETING OF THE AMERICAN INSTITUTE OF MINING ENGINEERS

Held at San Francisco, Cal., Thursday to Sunday, Sept. 16 to 19, 1915

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EXCURSIONS AND SOCIAL FUNCTIONS

The Institute joined with the American Society of Civil Engineers, the American Society of Mechanical Engineers, the American Institute of Electrical Engineers, and the Society of Naval Architects and Marine Engineers, in arranging for a special train from New York to San Francisco to carry members and guests of the five Societies, as well as foreign members of the International Engineering Congress. This train, consisting of Pullman cars and diners, left New York on the evening of Sept. 9. A stop of four hours was made on Sept. 10 for a visit to Niagara Falls. A stop of 10 hours was made at Colorado Springs where arrangements had been made for a trip by automobile through the famous Crystal Park, followed by trips to the summit of Pikes Peak, or visits to gold mills in Colorado Springs. Those members of the party who did not take these trips were taken by automobile to the Cripple Creek district where a visit was made to the mine and mill of Portland Gold Mining Co.

On Sept. 14 the special train stopped for 14 hours at the Grand Canyon, and then reached the City of San Francisco at 9:30 on the evening of Sept. 15. About 150 engineers and guests journeyed by this train and were unanimous in expressing appreciation of the arrangements that had been made by the Joint Committee having this in charge and by the Local Committee at Colorado Springs. Sincere appreciation was also expressed by the 30 members of the party who made the visit to the Cripple Creek district as guests of the Portland Gold Mining Co.

The Joint Committee on Arrangements of the five Societies was as follows:

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Excursions in San Francisco.—The Local Committee representing the National Engineering Societies and the International Engineering Congress had arranged a number of excursions in San Francisco for members of the various Societies involved, of which the following is a list:

Sept. 18: The San Francisco High Pressure Fire System, the Potrero Gas Works, and Electric Station "A," Pacific Gas & Electric Co.

Sept. 18: Spring Valley Water Works Properties on east side of San Francisco Bay.

Sept. 19: Spring Valley Water Works Reservoirs and Pumping Stations on San Francisco Peninsula.

Sept. 19: The Delta Lands of the Sacramento & San Joaquin Rivers.

Sept. 17-18: Great Western Power Co.'s Hydro-electric Development on Feather River and Dredging at Oroville.

Sept. 18-19: Pacific Gas & Electric Co.'s, Hydro-electric Development at Lake Spaulding and Drum Power House, and the Gold Mines at Grass Valley.

Sept. 17-19: Oil Fields at Coalinga.



BRONZE MEDAL PRESENTED TO THE INSTITUTE AT THE PANAMA-PACIFIC
INTERNATIONAL EXPOSITION

Boat Trip Around San Francisco Bay.—Through the courtesy of Colonel D. C. Jackling, the Local Committee of the Institute had provided for a boat trip around San Francisco Bay for members of the Institute and their guests only, on the beautiful and commodious yacht of Colonel Jackling, which holds comfortably 200 persons. Only 200 tickets were issued and every one of these was eagerly sought after. A buffet luncheon and afternoon tea were served on board the yacht; the trip included a visit of about an hour's duration to the smelting plant of the Selby Smelting & Lead Co. There does not exist a better opportunity for promoting acquaintance among a large number of people than by means of an attractive boat trip of this character and it is safe to say that every member of the party enjoyed himself or herself to the utmost. This trip started at 10:00 a.m. on Saturday, Sept. 17, and all the guests were on shore again by 4:30 in the afternoon.

Garden Party of Charles Butters.—On Sunday afternoon, Sept. 19, members and guests of the Institute were entertained at a garden party by Mr. and Mrs. Charles Butters in their beautiful place in Berkeley. This event proved to be one of the most enjoyable of the entire meeting.

Banquet.—A complimentary banquet was tendered to visiting members and guests by the San Francisco members on Friday evening, Sept. 17 at 7:30 p.m., at the Palace Hotel. This banquet was attended by about 400 persons. Mr. E. H. Benjamin acted as toastmaster and toasts were responded to by the following speakers:

BENJAMIN IDE WHEELER: An Engineer

W. L. SAUNDERS: California

VAN H. MANNING: Cooperation

GEORGE OTIS SMITH: The Outlook

T. A. RICKARD: The Ladies

Luncheons.—Thursday, Sept. 16, luncheon was had at Bellevue Hotel where the sessions were held. On Friday there was a joint subscription luncheon of the Institute and the American Electrochemical Society at the Clift Hotel, adjoining the Bellevue Hotel.

Entertainment of Ladies.—On Wednesday evening, Sept. 15, the Ladies' Committee held an informal reception for the visiting ladies and members in the lobby of the Bellevue Hotel. This reception began shortly after the arrival of the Engineers Special Train from the East. On Thursday, Sept. 16, the ladies were entertained by an automobile trip through the famous Golden Gate Park and the Presidio, after which a complimentary luncheon was served on the Exposition grounds; the ladies of San Francisco acted as guides to the visiting ladies in seeing the principal objects of interest in the Exposition. On Friday morning the ladies were taken through Chinatown, a favorite trip of all visitors to San Francisco.

AMERICAN INSTITUTE OF MINING ENGINEERS' DAY AT THE EXPOSITION

The management of the Panama-Pacific International Exposition had designated Friday, Sept. 17, as the American Institute of Mining Engineers' Day at the Exposition and in the afternoon at 4 o'clock ceremonies were held by the officers of the Exposition and the officers of the Institute in the Court of Abundance. These ceremonies consisted in the presentation to the Institute of a bronze medal (see accompanying illustrations). The presentation was made on behalf of the Exposition officers by Director F. L. Brown, in a felicitous speech, thanking the Institute for its share in the activities of the Exposition by holding its meeting in San Francisco

and joining in the support of the International Engineering Congress in San Francisco. President Saunders responded on behalf of the Institute in the following words:

It is said that the last words of William IV on hearing the sound of cannon on the anniversary of the Battle of Waterloo, were, "That was a great day for England;" and so as the members of this Institute look out upon this great exposition and see and hear what California, a mining State, has done in but little more than a century, we may perhaps be pardoned for exclaiming with joy and pride, This is a great day for the American Institute of Mining Engineers.

We are glad to be here, we are glad that we are alive: happy in the thrill of vitality which is quickened by California spirit and by the elixir of your glorious climate. On behalf of the Institute which I represent, I thank you for giving us this day. In thus honoring the Institute you are paying a tribute of regard and appreciation to the whole mining industry; to the lusty pioneer who blazed the trail and whose dry bones were the only monuments to mark the milestones of his progress: to the educated engineer and the capitalist, whose skill and money were spent beneath the soil in daring efforts to release the hidden forces of Nature and put them to the service of mankind. You are paying tribute to the geologist, the chemist, and the metallurgist, for they too are mining engineers and members of this Institute.

I speak of California as a mining State; as such were you born. You received your baptism in the waters of the South Fork of the American River, when Marshall discovered gold at Sutter's Mill in 1848. This was the beginning of the golden age for California and the world. The golden age means more than the age of gold; it is the age of progress, of prosperity and industrial renown and wealth; the age of railways and steamships, of manufacture, of the steam engine, of electricity, of the telephone, telegraph, the automobile and the aeroplane. All these things were made possible, yes, they were initiated and installed through means which were afforded by the mining industry. The mining engineer has multiplied and replenished the earth through subduing it; he has released those forces of nature which have been hidden through countless ages; he has through the science of metallurgy and chemistry made it economically possible to widen and enlarge to enormous proportions the use and influence of these forces. Look into the modern locomotive and steamship and we shall see that almost every pound of their vital structure is a product of the mines: The lathe that fashioned the shaft is itself a product of the mine and it gets its power to turn from coal mined by machines which are themselves products of the mines. Lest you say to all this that the art of mining is as old as the hills let us remind you of the fact that mining at low cost, mining low-grade ore, and doing these things on a large scale and the art of reduction and treatment through metallurgy, are the accomplishments of the mining engineer of our time. This was the key which unlocked those forces which have been the impulse behind the great industrial progress of the world. This too has all been accomplished since the discovery of gold in California.

Let us review some of the facts of history. The first half of the nineteenth century was a period marked by little progress in industrial wealth. The world's output of gold from 1800 to 1835 varied from \$10,000,000 to \$15,000,000 annually. The yearly product of iron during this period was only between 2,000,000 and 3,000,000 tons, coal less than 50,000,000 tons, copper less than 50,000,000 pounds. There were no transcontinental railways, no great steamships, no battleships, no factory system of manufacture, no large corporations. The total gold product of the United States during the period mentioned scarcely reached \$1,000,000 annually. Beginning about the middle of the century when California set the pace by adding in a single year \$50,000,000, and even \$80,000,000 to the gold of the world, there followed a steady and increasing progress made in the output of iron, coal, and copper. Railway building progressed on almost parallel lines, so too did the deposits in the banks of the United States. It is of interest to note here that the American Institute of Mining Engineers got aboard in 1871 and thereafter shared closely in growth and prosperity this march of industrial progress.

The world's production of gold at the present time is over \$450,000,000 annually. The United States alone now produces over 26,000,000 tons of pig iron annually, and about 550,000,000 tons of coal. The world's yearly production of copper is now over 900,000 tons and of this the United States produced over 500,000 tons. So too we find that the average per capita wealth of the people of the United States has risen from \$300 in 1850 to over \$1,300. The influence of mining on wealth is conspicuously shown in the case of your neighbor, the State of Nevada, where the per capita wealth of her people is about \$4,800. Nevada is strictly speak-

ing a mining State, with easy divorce laws to interest and allure the homeless and distracted mining engineer.

California has added more than \$1,600,000,000 to the gold supply of the world. Her record average of over \$25,000,000 in gold annually for 50 years is unprecedented. It has been estimated that the actual value of gold deposits should be multiplied eight times to get the true value in credit and capital: If this is true then California has in her gold supply alone contributed over \$12,000,000,000 to the capital wealth of the United States.

Gold is the standard of all values. It is the measure of credit, the basis of exchange. "Change and decay in all around I see," but in gold there is no change, no decay. The lure of gold has discovered continents and turned deserts into fertile lands. Gold was the beacon light which led Columbus across the Atlantic. Cortez, Pizarro, Balboa, and others who are called explorers were really pioneers and prospectors whose voyages led to the early development of mining in Peru and Chile. But not till California led the way did the mine explorer become a world builder; he was the advance agent of prosperity.

The stability of this country in peace and in war is due mainly to the mining engineer. Our industrial strength comes from the mines, the mills, and the furnaces which are now so well organized on a peace basis and which have in their substance all the sinews of war. Military supremacy comes from the same elements as industrial supremacy. It is mainly a question of organizing the resources of the country. Of one thing we may feel sure and that is that no nation can ever make a scrap of paper out of a gold eagle.

You may crush, you may shatter the coin if you will,
But the value of gold dust remains with it still.

It was because of the recognized importance of the mining and metallurgical industry that the American Institute of Mining Engineers was organized at Wilkes-Barre, Pa., in 1871. Following the War of the Rebellion and coincident with the development of the mining industry in the far Western States, there arose a great demand for mining engineers. It was recognized that the chief factor in the development of the western country was the mining industry; it created a romantic interest and it gave a stimulus to the people in general and even to Congress which granted large subsidies for railway building. Scientific schools were established and technical journals disseminated engineering literature; but the mining men, those actually engaged in the mines and mills and smelters, were not organized. They needed a common ground for the exchange of ideas, a forum for discussion and education; and so the Institute was established. It was the second society of its kind following the organization of the American Society of Civil Engineers. The *Transactions* of the Institute cover geology, assaying, milling, smelting, mining, quarrying, and general construction management. The papers submitted at the meetings cover a wide field and are recognized by engineers and scientists throughout the world as of significance and value.

The Institute is incorporated under the laws of the State of New York, organized and existing with the object of promoting the arts and sciences connected with the economic production of the useful minerals and metals and the welfare of those employed in these industries by means of meetings for social intercourse and the reading and discussion of professional papers, and to circulate by means of publications among its members the information thus obtained. Its membership includes those who have worked their way up from the ranks as well as graduates from scientific schools and colleges. Leading professors and technical educators have always been prominent in its membership. The welfare and safety of the mine workers is a subject of recent and earnest interest among the members, and a new field of usefulness has been developed in using the power and the influence of the Institute wherever the industry represented by mining and metallurgy or the people engaged in the work may be benefited through State or national legislation.

On behalf of the members of the American Institute of Mining Engineers, numbering now over 5,000 and scattered throughout the world, I extend to President Moore, to San Francisco, and to California, a full measure of warm congratulations and our best wishes for continued and increasing health, happiness, and prosperity.

TECHNICAL SESSIONS

The first technical session was held at 10:00 a.m. on Thursday, Sept. 16, 1915, in the Palm Room of Bellevue Hotel, Charles W. Merrill presiding.

An address of welcome on behalf of California was made by Mr. Arthur Arlett to whom Gov. Hiram Johnson deputed this duty. President Saunders replied on behalf of the Institute as follows:

On behalf of the American Institute of Mining Engineers it gives me pleasure to express our deep appreciation of the privilege extended to us of holding this meeting in California. We are glad to be here. It is always an inspiration and a pleasure to inhale your life-giving air. It is always a rare treat to get the true spirit of the West by rubbing elbows with your people.

But this meeting seems to me to have an importance, a far-reaching significance beyond any other that has been held since the foundation of the Institute. I am also impressed by the fact that it has a great significance to the people of California. In the busy march of your industrial life, with the many activities which now flourish engrossing your attention and giving wealth, prosperity and happiness to your people, does it occur to you that after all the basic foundation upon which your whole structure lies is the mining industry. Mining made California what she is and mining is still your greatest industry. Your mines have made it possible to establish the gold standard throughout the world. Your mines first created a human movement toward the far West which not only gave you those young and lusty pioneers of industry, but they peopled the western part of this country. It was the romantic mining spirit of California which influenced so successfully the building of railroads to the far West.

California created the mining engineer, and the mining engineer, true to his parent, has made California what she is. He did more than this, your scientific schools and your many workshops in the field sent forth the men who first developed the mining industry in America, Australia, Africa and other places of the world. Others have done much but the men of California blazed the first trail and did that part of the work which is the hardest of all, the work of the pioneer.

This Institute is the only organized body which for 44 years has represented the mining men, from the man with the pick, who rises to a higher charge, to the college man, the geologist, the chemist, the assayer, the engineer, the metallurgist and the manager. Our work is coördinated for the safety, welfare and profit of the industry.

We feel that we have a right to be here at this festival held in honor of your prosperity, and as we have shared with you some of the burdens and hardships of the past, you will I trust permit us at this happy time also to share with you at least some measure of your pride and rejoicing.

The following papers were then presented by their authors or authors' representatives:

Underground Mining Systems of Ray Consolidated Copper Co. By Lester A. Blackner. (Discussed by Sidney J. Jennings.)

Some Problems in Copper Leaching. By L. D. Ricketts. (Discussed by Frederick Laist, T. T. Read, E. H. Hamilton.)

Notes on Homestake Metallurgy. By Allan J. Clark. (Discussed by H. G. Clevenger, J. A. Fulton.)

The Metallurgy of Gold in the Witwatersrand District, South Africa. By F. L. Bosqui. (Discussed by Sidney J. Jennings, C. W. Merrill, Charles Butters, Hennen Jennings, Allan J. Clark and written discussion by W. A. Caldecott.)

On the afternoon of Thursday, Sept. 16, 1915, at 2:00 p.m., a Session on Gold and Silver was held in the Palm Room of Bellevue Hotel, F. Lynwood Garrison presiding.

The following papers were then presented by their authors or authors' representatives:

Mill and Cyanide Plant of Chiksan Mines, Korea. By Charles W. De Witt.

Slime Agitation and Solution Replacement Methods, West End Mill, Tonopah, Nev. By Jay A. Carpenter.

- Cyaniding Practice of Churchill Mining Co., Wonder, Nev. By E. E. Carpenter.
 The Tonopah Plant of the Belmont Milling Co. By A. H. Jones.
 A Rule Governing Cupellation Losses. By W. J. Sharwood. (Written discussion by Frederic P. Dewey.)
 Zinc-Dust Precipitation Tests. By Nathaniel Herz.
 Recovery of Mercury from Amalgamation Tailing, Buffalo Mines, Cobalt. By E. B. Thornhill. (Discussed by D. B. Huntley, H. G. S. Anderson.)

The following papers were read by title only:

- Amalgamation Tests. By W. J. Sharwood.
 Electric Furnace for Gold Refining at the Alaska-Treadwell Cyanide Plant. By W. P. Lass. (Written discussion by R. M. Keeney.)

On the afternoon of Thursday, Sept. 16, 1915, at 2:00 p. m., a Session on Geology and Mineralogy was held in the Red Room of Bellevue Hotel, Andrew C. Lawson presiding.

The following papers were then presented by their authors or authors' representatives:

- The Geology of the Iron-Ore Deposits in and Near Daquiri, Cuba. By James F. Kemp. (Discussed by W. E. Pratt.)
 The Occurrence of Covellite at Butte, Mont. By A. Perry Thompson. (Discussed by A. S. Eakle, H. W. Turner, C. F. Tolman, A. F. Rogers, L. C. Graton, A. C. Lawson, J. C. Ray.)
 The Formation and Distribution of Bog Iron Ores. By C. L. Dake. (Discussed by A. C. Lawson.)
 Geology of the Burro Mountains Copper District, New Mexico. By R. E. Somers. (Discussed by J. B. Umpleby, L. C. Graton.)
 Method of Making Mineralogical Analysis of Sand. By C. W. Tomlinson.
 The Copper Deposits of San Cristobal, Santo Domingo. By Thomas F. Donnelly. (Written discussion by F. Lynwood Garrison.)

The following papers were read by title only:

- Additional Data on Origin of Lateritic Iron Ores of Eastern Cuba. By C. K. Leith and W. J. Mead.
 The Formation of Oxidized Ores of Zinc from the Sulphide. By Y. T. Wang.
 The Formation and Distribution of Residual Iron Ores. By C. L. Dake.

On Friday, Sept. 17, 1915, at 9:00 a. m., a Session on Petroleum and Gas was held in the Red Room of Bellevue Hotel, Arthur F. L. Bell presiding.

The following papers were then presented by their authors or authors' representatives:

- Petroleum as Fuel Under Boilers and in Furnaces for Heating, Melting, and Heat Treatment of Metals. By W. N. Best. (Discussed by A. F. L. Bell, William A. Williams, Hennen Jennings, D. T. Day, Mark L. Requa.)
 Gasoline from "Synthetic" Crude Oil. By Walter O. Snelling. (Discussed by A. C. McLaughlin, William A. Williams, W. N. Best, Mark L. Requa, A. F. L. Bell, D. T. Day, Walter Stalder.)
 The Possible Occurrence of Oil and Gas Fields in Washington. By Charles E. Weaver. (Discussed by A. C. McLaughlin, W. N. Best, Milnor Roberts, W. E. Pratt, D. T. Day, George O. Smith.)
 Protecting California Oil Fields from Damage by Infiltrating Water. By R. P. McLaughlin. (Discussed by A. C. McLaughlin, M. E. Lombardi, A. F. L. Bell, William A. Williams, M. L. Requa, C. D. Keen.)
 The Cost of Maintaining Production in California Oil Fields. By M. E. Lombardi. (Discussed by M. L. Requa, C. D. Keen, H. M. Bacon.)
 Important Topping Plants of California. By Arthur F. L. Bell. (Discussed by William A. Williams, D. T. Day.)

The following papers were presented by title only:

- The Occurrences of Petroleum in the Oil Fields of Eastern Mexico as Contrasted with those of Texas and Louisiana. By E. T. Dumble.
 Oil, Gas and Water Content of Dakota Sand in Canada and United States. By L. G. Huntley.

Correlation and Geological Structure of the Alberta Oil Fields. By D. B. Dowling.
 The Mexican Oil Fields. By L. G. Huntley.
 Sliding Royalties for Oil and Gas Wells. By Roswell H. Johnson. (Discussed by
 William A. Williams, A. C. McLaughlin, A. F. L. Bell.)
 The Furbero Oil Field, Mexico. By E. DeGolyer.

On Friday, Sept. 17, 1915, at 9:00 a.m., Session on Electro-Metallurgy.
 Joint Session with the American Electrochemical Society, in the Palm
 Room of Bellevue Hotel, Lawrence W. Addicks, President of the American
 Electrochemical Society, presiding.

The following papers were presented by their authors or authors' representatives:

The Thermal Insulation of High-Temperature Equipment. By Percy A. Boeck.
 (Discussed by Lawrence Addicks, J. W. Beckman, A. H. Kreiger, J. W. Richards,
 Kenneth Seaver.)
 Metallurgical Industries as Possible Consumers of Electric Power. By Dorsey A.
 Lyon and Robert M. Keeney. (Discussed by Lawrence Addicks, G. H. Clevenger,
 C. H. Booth, Kenneth Seaver, J. W. Richards, J. W. Beckman, T. T. Read.)
 The Electric Precipitation of Gold, Silver and Copper from Cyanide Solutions. By
 G. H. Clevenger. (Discussed by Dr. Daschak, J. W. Richards, T. T. Read.)
 Solution Stratification as an Aid to the Purification of Electrolytes. By Francis R.
 Pyne. (Discussed by Frederick Laist, Lawrence Addicks, J. W. Richards.)
 Roasting and Leaching Concentrator Slimes Tailings. By Lawrence Addicks.
 (Discussed by Frederick Laist, Lawrence Addicks, J. W. Richards.)
 The Electrolysis of Copper Sulphate Liquors, Using Carbon Anodes. By Lawrence
 Addicks. (Discussed by L. D. Ricketts, E. H. Hamilton, Frederick Laist,
 Charles Butters.)
 Hydro-Electrolytic Treatment of Copper Ores. By Robert Rhea Goodrich.

The following papers were read by title only:

Melting of Ferro-Alloys in the Electric Furnace. By R. S. Wile.
 Radiography of Metals. By Wheeler P. Davey.

On Friday afternoon, Sept. 17, 1915, at 1 o'clock, a Session on Mining,
 Milling and Non-Ferrous Metallurgy, was held in the Palm Room of
 Bellevue Hotel, Karl Eilers presiding.

The following papers were presented by their authors or authors' representatives:

Tramming and Hoisting at the Copper Queen Mine. By Gerald Sherman.
 Ventilation of the Copper Queen Mine. By Charles A. Mitke. (Discussed by
 Gerald Sherman, J. P. Hodgson.)
 Fire-Fighting Methods at Mountain View Mine, Butte, Mont. By C. L. Berrien.
 Standardizing Rock-Crushing Tests. By M. K. Rodgers.
 The Concentrator of the Timber Milling Co., Butte, Mont. By Theodore Simons.
 The British Columbia Copper Co.'s Smelter, Greenwood, B. C. By Frederic K.
 Brunton.
 The Advantages of High-Lime Slags in the Smelting of Lead Ores. By S. E. Bretherton.
 (Written discussion by A. Eilers.)

The following papers were read by title only:

The Application of the Apex Law at Wardner, Idaho. By Fred T. Greene.
 Mine Pumping. By Charles Legrand.
 Mining Conditions on the Witwatersrand. By W. L. Honnold.
 Churn-Drilling Costs, Sacramento Hill. By Arthur Notman.
 The Stresses in the Mine Roof. By R. Dawson Hall.
 Kick vs. Rittinger. By Arthur O. Gates. (Written discussion by Algernon Del Mar.)
 Rotary Desulphurizing Kilns. By S. E. Doak.
 Hardinge Mill Data. By Arthur F. Taggart.
 The Salda Smelter. By F. D. Weeks.
 Lead Smelting at El Paso. By H. F. Easter.
 The Mellen Rod-Casting Machine. By R. C. Patterson, Jr.
 Conveyor-Belt Calculating Chart. By J. D. Mooney and D. L. Darnell.

On Friday afternoon, Sept. 17, 1915, at 1 o'clock, a Session on Iron and Steel was held in the Red Room of Bellevue Hotel, Joseph W. Richards presiding.

The following papers were presented by their authors or authors' representatives:

- Manufacture and Tests of Silica Coke-oven Brick. By Kenneth Seaver. (Discussed by William A. Williams, J. W. Richards.)
 Iron Ores of California and Possibilities of Smelting. By C. Colcock Jones. (Discussed by D. A. Lyon, J. W. Booth, Bradley Stoughton, H. W. Lash, J. W. Richards.)
 The Electric Furnace in the Foundry. By William G. Kranz. (Written discussion by M. Petinot.) (Discussed by J. W. Booth, J. W. Richards, H. W. Lash.)
 Commercial Production of Sound Homogeneous Steel Ingots and Blooms. By E. Gathmann. (Discussed by H. W. Lash, J. W. Richards.)

The following papers were read by title only:

- Conversion Scale for Centigrade and Fahrenheit Temperatures. By Hugh P. Tieman. (Discussed by J. W. Richards.)
 The Duplex Process of Steel Manufacture at the Maryland Steel Works. By F. F. Lines.
 Suggestions Regarding the Determination of the Properties of Steel. By A. N. Mitinsky. (Written discussion by H. M. Howe, Mansfield Merriman, Gaetano Lanza, Lawford H. Fry, H. V. Wille, James E. Howard, George K. Burgess.)

NAVAL CONSULTING BOARD

The first meeting of the Naval Consulting Board was held at the office of the Secretary of the Navy, Washington, D. C., on Oct. 6, 1915. The Board organized by electing Thomas A. Edison, Chairman, Peter Cooper Hewitt, First Vice-Chairman, William L. Saunders, Second Vice-Chairman, and Thomas Robins, Secretary. All of these officers are members of the American Institute of Mining Engineers. The other members of the Board are given below, and those who are members of the Institute are marked with an asterisk:

*W. R. WHITNEY
 L. H. BAEKELAND.
 FRANK JULIAN SPRAGUE.
 BENJAMIN C. LAMME.
 R. S. WOODWARD.
 ARTHUR GORDON WEBSTER.
 A. M. HUNT.
 ALFRED CRAVEN.
 *SPENCER MILLER.
 WILLIAM LE ROY EMMETT.

MATTHEW B. SELLERS.
 HUDSON MAXIM.
 HOWARD T. COFFIN.
 ANDREW J. RIKER.
 HENRY A. WISE WOOD.
 ELMER A. SPERRY.
 *BENJAMIN B. THAYER.
 *J. W. RICHARDS.
 *LAWRENCE ADDICKS.

At this first meeting an address was made by the Secretary of the Navy Honorable Josephus Daniels, in which he outlined the needs of the Navy Department, mentioning especially a better type of motor for submarines and airships. Secretary Daniels said that the chief manner in which the Naval Consulting Board could be of service to the Navy Department was to form a clearing house for ideas and suggestions; to give expert advice in big problems with which the navy is confronted; to initiate suggestions which will add to the efficiency of the navy; and to suggest ways in which its expert knowledge may be employed. The Secretary of the Navy also asked the Board to decide whether or not Congress should be requested to appropriate a sufficient sum of money to provide a laboratory for research and experiment. To this request the Board promptly responded

by recommending a plan to establish a research laboratory to cost eventually \$5,000,000, in which actual tests may be made of all seemingly meritorious naval inventions. The secretary of the Navy responded that he would at once ask Congress for an appropriation of \$1,000,000 to begin this work. Following the meeting at the Navy Department the members of the Consulting Board were addressed at the White House by the President of the United States, who congratulated them upon beginning their work and announced himself in favor of adequate preparedness on the part of the United States.

The Board then proceeded on the presidential yacht, *Mayflower*, to the Indian Head gun-proving grounds, where various tests were exhibited and explained to the Board.

The meeting on Oct. 6, 1915, was protracted until nearly midnight in a discussion of various matters before the Board.

The second meeting of the Board was held in New York on Nov. 4, 1915. In the absence of Mr. Edison, W. L. Saunders presided. In accordance with the policy that had been determined upon at the first meeting, the Board divided itself into the following committees for the better carrying out of the work:

Chemistry and Physics.—Lawrence Addicks, L. H. Baekeland, Joseph W. Richards, M. B. Sellers, A. G. Webster, R. S. Woodward, and W. R. Whitney, *Chairman*.

Aeronautics (including Aero Motors).—Howard E. Coffin, P. C. Hewitt, Andrew L. Riker, M. B. Sellers, E. A. Sperry, A. G. Webster, and Henry A. Wise Wood, *Chairman*.

Internal Combustion Motors.—Howard E. Coffin, M. B. Sellers, E. A. Sperry, and Andrew L. Riker, *Chairman*.

Electricity.—Lawrence Addicks, William Le Roy Emmet, P. C. Hewitt, B. G. Lamme, A. G. Webster, and Frank J. Sprague, *Chairman*.

Mines and Torpedoes.—L. H. Baekeland, M. R. Hutchinson, Hudson Maxim, and Elmer A. Sperry, *Chairman*.

Submarines.—A. M. Hunt, M. R. Hutchinson, W. L. Saunders, Frank J. Sprague, and William Le Roy Emmet, *Chairman*.

Ordnance and Explosives.—L. H. Baekeland, A. M. Hunt, M. R. Hutchinson, Frank J. Sprague, A. G. Webster, W. R. Whitney, Henry A. Wise Wood, R. S. Woodward, and Hudson Maxim, *Chairman*.

Wireless and Communications.—A. G. Webster, W. R. Whitney, and P. C. Hewitt, *Chairman*.

Transportation.—Howard E. Coffin, Alfred Craven, Spencer Miller, A. L. Riker, Thomas Robins, W. L. Saunders, Henry A. Wise Wood, and Benjamin B. Thayer, *Chairman*.

Production, Organization, Manufacture, and Standardization.—Lawrence Addicks, William Le Roy Emmet, B. G. Lamme, Thomas Robins, W. L. Saunders, Benjamin B. Thayer, and Howard E. Coffin, *Chairman*.

Ship Construction.—Spencer Miller, Joseph W. Richards, Henry A. Wise Wood, and Frank J. Sprague, *Chairman*.

Steam Engineering and Ship Propulsion.—William Le Roy Emmet, B. G. Lamme, Joseph W. Richards, M. B. Sellers, and Andrew M. Hunt, *Chairman*.

Life-Saving Apparatus.—Hudson Maxim, Thomas Robins, and Spencer Miller, *Chairman*.

Aids to Navigation.—Alfred Craven, A. M. Hunt, Henry A. Wise Wood, D. S. Woodward, and Elmer A. Sperry, *Chairman*.

Food and Sanitation.—Hudson Maxim, Benjamin B. Thayer, W. R. Whitney, R. S. Woodward, and L. H. Baekeland, *Chairman*.

Public Works, Yards, and Docks.—Lawrence Addicks, A. M. Hunt, Spencer Miller, Joseph W. Richards, and Alfred Craven, *Chairman*.

Remember the New York Meeting, Feb, 14 to 17, 1916.

THE ENGINEERING FOUNDATION

The members of the Institute will recall the account given in the March *Bulletin* of the inauguration exercises of the Engineering Foundation which was inhibited by an initial gift of \$200,000 by Mr. Ambrose Swasey, to be administered by the United Engineering Society as trustees.

The Engineering Foundation Board is now constituted as follows:

GANO DUNN, *Chairman*,
 EDWARD D. ADAMS, *Vice-Chairman*
 FREDERICK R. HUTTON, *Secretary*,
 JOSEPH STRUTHERS, *Treasurer*,
American Society of Civil Engineers.
 J. WALDO SMITH,
 CHARLES WARREN HUNT.
American Society of Mechanical Engineers.
 JESSE M. SMITH,
 ALEX. C. HUMPHREYS.
American Institute of Electrical Engineers.
 M. I. PUPIN,
 CHARLES E. SCRIBNER.
American Institute of Mining Engineers.
 ALBERT R. LEDOUX,
 BENJAMIN B. THAYER.
Members at Large.
 EDWARD DEAN ADAMS,
 HOWARD ELLIOTT.

The organization meeting was held in the early summer and plans made for carrying out the work of the Board. Another meeting is planned for the early future and, as soon as material is ready for publication, notice will be made in this *Bulletin*.

INTERNATIONAL ENGINEERING CONGRESS

The papers presented at the International Congress, held in San Francisco, Cal., Sept. 20 to 25, 1915, are published in 11 volumes of *Transactions*. In addition, an index volume is published, containing besides the author and title index, a report of the business meetings of the Congress, and a brief digest of each paper presented.

The following is a schedule of the volumes:

Index and Digest.—General proceedings, indexes and digests.

Vol. I. Panama Canal.

II. Waterways and Irrigation.

III. Municipal Engineering.

IV. Railway Engineering.

V. Materials of Engineering Construction.

VI. Mechanical Engineering.

VII. Electrical Engineering and Hydro-electric Power Development.

VIII. Mining Engineering.

IX. Metallurgy.

X. Naval Architecture and Marine Engineering.

XI. Miscellaneous. Aeronautics; Refrigeration; Agricultural Engineering; Engineering Education; Heating and Ventilation.

Remember the New York Meeting, Feb. 14 to 17, 1916.

The complete list of contents of Vols. VIII and IX, which are of particular interest to the members of the American Institute of Mining Engineers, is given below.

Volume VIII

Mining Engineering

- Economic and Social Influence of Mining. W. H. Shockley, Palo Alto, Cal.
 Valuation of Metal Mines and Prospects. T. A. Rickard, San Francisco, Cal.
 The Valuation of Oil Lands and Properties. M. E. Lombardi, San Francisco, Cal.
 Valuation of Anthracite Mines. R. V. Norris, Wilkes-Barre, Pa.
 Valuation of Coal Lands. Samuel A. Taylor, Pittsburgh, Pa.
 Evaluating Coal Properties in Western Canada. R. W. Coulthard, Calgary, Alberta, Canada.
 Status of Coal Mines in France. E. Gruner, Comité des Charbonnages, Paris, France.
 Workmen's Compensation and Mine Safety. W. H. Wilson, Pittsburgh, Pa.
 Functions and Work of Exploration and Development Companies. H. W. Turner, San Francisco, Cal.
 The Financing of Mines in the United States. Luicis W. Mayer, New York, N. Y.
 European Mining Finance. J. L. Gallard, London, England.
 Organization and Staff of Mining Companies. W. H. Shockley, Palo Alto, Cal. and R. E. Cranston, San Francisco, Cal.
 Relation of Governments to Mining. Horace V. Winchell, Minneapolis, Minn.
 Mine Inspection. J. W. Paul, Pittsburgh, Pa.

Volume IX

Metallurgy

- Symposium on Iron and Steel. Henry M. Howe, Bedford Hills, N. Y.
 Iron and Steel Castings. John Howe Hall, High Bridge, N. J.
 The Duplex Process of Steel Manufacture. F. F. Lines, Sparrows Point, Md.
 Steel Making in the Electric Furnace. James H. Gray, New York, N. Y.
 Methods of Preventing Piping in Steel Ingots. Emil Gathmann, Baltimore, Md.
 Steel Alloys. George L. Norris, Pittsburgh, Pa.
 Case Hardening of Steel. Prof. F. Giolitti, Turin, Italy.
 Metallography and the Hardening of Steel. Albert Sauveur, Cambridge, Mass.
 Symposium on Copper. E. P. Mathewson, Anaconda, Mont.
 Progress in Copper Metallurgy. Thomas T. Read, San Francisco, Cal.
 Metallurgy of Copper in Japan. R. Kondo, Tokyo, Japan.
 Copper Metallurgy of the Southwest. James Douglas, New York, N. Y.
 Advances made in the Metallurgy of Copper, Globe District, Arizona. L. O. Howard, Globe, Ariz.
 Advances in Copper Smelting. Frederick Laist, Anaconda, Mont.
 Improvements in Design and Construction of Modern Copper Plants. Charles H. Repath, Los Angeles, Cal.
 Reduction Works. Copper Queen Consolidated Mining Company, Douglas, Arizona.
 Forest Rutherford, Douglas, Ariz.
 The Development of Electrolytic Copper Refining. Lawrence Addicks, New York, N. Y.
 Electrolytic Refined Copper. A. C. Clark, Raritan Copper Works, Perth Amboy, N. J.
 Leaching Copper Ores. W. L. Austin, Riverside, Cal.
 Physical Properties of Copper. Carle R. Hayward, Boston, Mass.
 Boronized Cast Copper. E. Weintraub, West Lynn, Mass.
 The Metallography of Copper. William Campbell, New York, N. Y.
 Symposium on Gold and Silver. C. W. Merrill, San Francisco, Cal.
 Coarse Crushing Plant; 1,000 tons Capacity. G. O. Bradley, San Francisco, Cal.
 Crushing and Grinding. L. D. Mills, and M. H. Kuryla, San Francisco, Cal.
 Solution of Gold and Silver. M. H. Kuryla, San Francisco, Cal.
 Filtration or Separation of Metal Bearing Solution from Slime Residue. L. D. Mills, San Francisco, Cal.
 Precipitation. G. H. Clevenger, University, Cal.
 The Smelting and Refining of Lead. H. O. Hofman, Boston, Mass.
 Symposium on the Metallurgy of Zinc. Walter Ranton Ingalls, New York, N. Y.

Some Main Points in the Economics of the Metallurgy of Zinc. Walter Renton Ingalls, New York, N. Y.
 The Development of Zinc Smelting in the United States. George C. Stone, New York, N. Y.
 Ore Dressing. Robert H. Richards, Boston, Mass.
 Electro-Metallurgy. E. F. Roeber, New York, N. Y.
 Metallography and Technology of Non-Ferrous Alloys. William Campbell, New York, N. Y.
 Symposium on the Utilization of Fuels in Metallurgy. C. H. Fulton, Cleveland, Ohio.
 Pulverized Coal in Reverberatory Furnaces. D. H. Browne, New York, N. Y.
 Burning Pulverized Coal in Copper Reverberatories. E. P. Mathewson, Anaconda, Mont.
 Gas Producer Development. Z. C. Kline, Central Falls, R. I.
 Surface Combustion (What is it?). C. E. Lucke, New York, N. Y.

A Fee of \$5 entitles to Index Volume and any *one* other Volume of the *Transactions*.

Extra volumes of the *Transactions* may be obtained by subscribers for the following prices, which include delivery to any point in the Postal Union:

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The purchase of nine extra volumes, in addition to the Index and the volume obtained through the subscription fee, secures a full set of the Congress publications for \$27.50 for paper binding or \$30 for cloth. These prices do not apply to duplicate copies of any one volume. Special prices will be quoted for the sale of several copies of any one volume.

The subscription list will be kept open until the latter part of December, 1915. All correspondence in regard to subscriptions and extra volumes should be addressed to the International Engineering Congress, Foxcroft Building, San Francisco, Cal.

SECOND PAN-AMERICAN SCIENTIFIC CONGRESS

The Second Pan-American Scientific Congress will be held under the auspices of the United States Government in Washington, D. C., Dec. 27, 1915, to Jan. 8, 1916. The organization officers are John Barrett, LL. D., Director General of the Pan-American Union, Secretary General; Glen Levin Swiggett, Ph. D., Assistant Secretary General; the headquarters of the Congress are the Pan-American Union, Washington, D. C. The program of the Congress is divided into nine main Sections, which are given below with the name of the Chairman of each Section:

1. Anthropology, William H. Holmes, B. S., Smithsonian Institution, Washington.
2. Astronomy, Meteorology, and Seismology, Robert S. Woodward, Ph. D., Carnegie Institution, Washington, D. C.
3. Conservation of National Resources, Agriculture, Irrigation and Forestry, George M. Rommel, B. S., Bureau of Animal Industry, Department of Agriculture, Washington, D. C.
4. Education, P. P. Claxton, LL. D., Bureau of Education, Washington, D. C.

5. Engineering, W. H. Bixby, Brig. General U. S. A., Retired, Washington, D. C.
6. International Law, Public Law, and Jurisprudence, James Brown Scott, A. M., J. U. D., LL. D., Carnegie Endowment for International Peace, Washington, D. C.
7. Mining and Metallurgy, Economic Geology, and Applied Chemistry, Hennen Jennings, C. E., Washington, D. C.
8. Public Health and Medical Science, Wm. C. Gorgas, M. D., Sc. D., Surgeon General U. S. A., Washington, D. C.
9. Transportation, Commerce, Finance, and Taxation, L. S. Rowe, Ph. D., President American Academy of Political and Social Science, Philadelphia, Pa.

Each Section is further divided into Sub-Sections, of which there are 45 in all, each one having a special committee and program. There will be general sessions of the Congress as a whole, and also sessions of each of the Sub-Sections, as well as certain joint sessions in which two or more Sub-Sections will meet together. There will also be joint sessions between certain Sections and Sub-Sections of the Congress with National Associations meeting in Washington at the time of the Congress.

The following persons will be members of the Congress:

The official delegates of the countries represented.

The representatives of the universities, institutes, societies, and scientific bodies of the countries represented.

Such persons in the countries participating in the Congress as may be invited by the Executive Committee, with the approval of the countries represented.

All writers of papers.

All members of the Congress shall be entitled to attend its sessions, to take part in the debates and to receive a copy of such publications as the Executive Committee may issue. There will be no membership fee of any character.

Up to the present time twenty countries in North and South America have joined in the Congress and appointed delegates to attend. The members of this Institute are naturally most interested in the Section on Mining, Metallurgy, Economic Geology and Applied Chemistry, and the officers and members of these Committees are as follows:

Mining

- Van. H. Manning, Director of the U. S. Bureau of Mines, Washington, D. C., *Chairman*.
 J. F. Callbreath, Secretary of the American Mining Congress, Washington, D. C.
 C. H. Lindley, Lawyer, Authority on Mining Law, San Francisco, Cal.
 E. W. Parker, Statistician, Mineral Resources of the United States, U. S. Geological Survey, Washington, D. C.
 H. C. Perkins, Mining Engineer, Washington, D. C.
 G. S. Rice, Chief Mining Engineer of the U. S. Bureau of Mines, Pittsburgh, Pa.
 W. L. Saunders, Engineer, President of the American Institute of Mining Engineers, New York, N. Y.
 B. B. Thayer, Past President of the American Institute of Mining Engineers, New York, N. Y.

Metallurgy

- W. R. Ingalls, President of the Mining and Metallurgical Society of America, and Editor of the Engineering and Mining Journal, New York, N. Y., *Chairman*.
 F. G. Cottrell, Chief Chemist of the U. S. Bureau of Mines, San Francisco, Cal.
 R. H. Richards, Professor Emeritus of Mining Engineering and Metallurgy, Massachusetts Institute of Technology, Boston, Mass., former President of the American Institute of Mining Engineers.
 Bradley Stoughton, Metallurgical Engineer, Secretary of the American Institute of Mining Engineers, New York, N. Y.
 L. D. Ricketts, Mining and Metallurgical Engineer, New York, N. Y.

Karl Eilers, Metallurgical Engineer, New York, N. Y.
 W. R. Walker, Metallurgist, New York, N. Y.
 G. H. Clevenger, Professor of Metallurgy, Stanford University, Cal.

Economic Geology

George Otis Smith, Director of the U. S. Geological Survey, Washington, D. C., *Chairman*.
 J. C. Branner, President of Stanford University, Stanford University, Cal.; former President of the Geological Society of America.
 J. F. Kemp, Professor of Geology, Columbia University, New York, N. Y., former President of the American Institute of Mining Engineers.
 Waldemar Lindgren, Professor of Economic Geology, Massachusetts Institute of Technology, Boston, Mass.
 C. R. Van Hise, President of the University of Wisconsin, Madison, Wis., former President of the Geological Society of America.
 David White, Chief Geologist of the U. S. Geological Survey, Washington, D. C.
 I. C. White, State Geologist of West Virginia, Morgantown, W. Va., formerly Chief of the Brazilian Coal Commission.
 Bailey Willis, Consulting Geologist to Argentine Government; Geologist, U. S. Geological Survey, Washington, D. C.

Applied Chemistry

Charles E. Monroe, Authority on Explosives; Dean of Graduate Studies, George Washington University, Washington, D. C.; former President of the American Chemical Society, *Chairman*.
 Carl L. Alsberg, Chief of the Bureau of Chemistry, Department of Agriculture, Washington, D. C.
 C. H. Herty, President of the American Chemical Society, Chapel Hill, N. C.
 B. C. Hesse, Chemist, New York, N. Y.
 W. F. Hillebrand, Chief Chemist of the U. S. Bureau of Standards, Washington, D. C., former President of the American Chemical Society.
 F. A. Lidbury, Electrochemist; President of the American Electrochemical Society, Niagara Falls, N. Y.
 P. C. McIlhenny, Chemist, Chairman of the New York Section, Society of Chemical Industry, New York, N. Y.
 Harvey W. Wiley, Chemist; former President of the American Chemical Society, Washington, D. C.

The program of the session on Mining and Metallurgy will include a discussion of the following subjects:

I. Mining laws and regulations, and accident statistics.

- (a) *Codification* of mining laws, with the purpose of assembling: (1) the mining statutes of each country with amendments to date; (2) State laws and regulations; and (3) the legal interpretation of these laws, including both legal definitions of mining terms and the court decisions defining the scope of applicability of particular statutes.
- (b) *Methods of determining fatality rates* in mining and other mineral industries, with reference to the adoption of uniform methods.
- (c) *Methods of classifying* mining and other industrial accidents with a view to the adoption of uniform methods.

II. Methods of mining and treating minerals, with reference to increased economy and efficiency and greater safety.

(a) *Metal mining* (surface and underground)—

1. Platinum, gold, silver, and other noble metal ores.
2. Copper, tin, lead, and zinc ores.
3. Iron and manganese ores.
4. Ores of vanadium and other rare metals.

(b) *Non-metal mining* (surface and underground), including oil, gas, and salt wells.

1. Fuels.
 - a. Coal, lignite, peat.
 - b. Oil and natural gas.
2. Asphalt and other hydrocarbons not used as fuel.
3. Sodium chloride, sodium nitrate, phosphates.
4. Building stones, clays, cement materials.
5. Precious stones.
6. Other minerals.

III. Mining costs.

- (a) Costs of acquiring title to mineral lands or holding mineral leases; royalties; taxes.
- (b) Costs of mining, underground and surface, with reference to labor supply and greater use of water power, steam, electricity, and labor-saving machinery.
- (c) Costs of ore dressing, with reference to labor supply and increased use of labor-saving devices.
- (d) Transportation costs, with reference to present methods of transportation and the lowering of cost to be effected by better roads, and greater use of aerial conveyors and steam or electric railroads.

IV. Statistics of cost data on the use of electricity in mining, including also discussion on the development of water power especially for this purpose.

V. Hydro-metallurgy.

- (a) Treatment of the precious metals, including especially recent developments in the cyanide process.
- (b) Extraction of copper from its ores. (So great developments have recently taken place in this phase of metallurgy that we may say that the whole copper industry seems to be on the verge of a great revolution.)
- (c) Extraction of zinc and other metals by wet methods.
- (d) The cost of hydro-metallurgical processes as compared with smelting processes.

VI. Electro-metallurgy.

- (a) The electrolysis of copper, lead, nickel, and other metals.
- (b) Electric smelting of ores of iron, copper, nickel, and zinc.
- (c) Electric refining of steel.
- (d) The use of electric melting in steel, iron, and brass foundries.
- (e) The cost of electric processes as compared with fuel processes under conditions of costly fuel.

VII. Fuels.

- (a) The preparation of pulverized solid fuels of good grade from inferior and dirty coal, lignite, etc.
- (b) The use of pulverized solid fuel in reverberatory furnaces for melting, smelting, annealing, etc., including its use in the copper, iron, steel, and zinc industries.
- (c) The use of oil in reverberatory furnaces, with and without preheated air, for melting, smelting, annealing, etc., including its use in the iron, steel, copper, and zinc industries.
- (d) The use of oil and pulverized solid fuel in special types of furnace.
- (e) The cost of using oil and pulverized fuels as compared with coal and manufactured gas.

The program of the session on Economic Geology and Mineral Resources of the different countries will consist of reports and papers descriptive of the mineral deposits, both developed and undeveloped, in the different countries.

The conduct of the Congress is entirely under the control of the Department of State of the United States and the Executive Committee consists of the following persons:

WILLIAM PHILLIPS, A. B., Third Assistant Secretary of State, *Chairman ex officio*.

JAMES BROWN SCOTT, J. U. D., Secretary, Carnegie Endowment for International Peace, *Vice-Chairman*.

WILLIAM H. WELCH, M. D., LL. D., President, National Academy of Sciences, *Honorary Vice-Chairman*.

JOHN BARRETT, LL. D., Director General, Pan-American Union.

W. H. BIXBY, Brigadier General, U. S. A., Retired.

PHILANDER F. CLAXTON, LL. D., Commissioner of Education.

WILLIAM C. GORGAS, M. D., Sc. D., Surgeon General, U. S. A.

WILLIAM H. HOLMES, B. S., Head Curator, Smithsonian Institution.

HENNER JENNINGS, C. E., former President, London Institution Mining and Metallurgy.

GEORGE M. ROMMEL, B. S., Chief, Animal Husbandry Division, Bureau of Animal Industry, Department of Agriculture.

L. S. ROWE, Ph. D., President American Academy of Political and Social Science.

ROBERT S. WOODWARD, Ph. D., President, Carnegie Institution of Washington.

PERSONAL

(Members are urged to send in for this column any notes of interest concerning themselves or their fellow-members.)

Members and guests who registered at Institute headquarters during the period Oct. to Nov. 10, 1915:

Lester S. Thompson, New York, N. Y.

Arthur L. Sweetser, Boston, Mass.

William Reed, El Oro, Mexico.

Dorsey A. Lyon, Salt Lake City, Utah

Heinrich Ries, Ithaca, N. Y.

D. W. Blaylock has resigned his position with the St. Joseph Lead Co., Bonne Terre, Mo., and has accepted the position of division engineer for the Big Creek Coal Co., of Chicago, at Harrisburg, Ill.

R. B. Lamb announces the removal of his office to Room 1809, 43 Exchange Place, New York, N. Y.

C. C. Sanders has taken a bond and lease on the Red Mountain group of 34 mining claims (copper), Patagonia, Ariz., and has commenced extensive development.

Charles E. Addams, formerly superintendent of the Ray Consolidated Copper Co., has been appointed manager of the Ray Hercules Co.

Dr. Bailey Willis has been appointed head of the Department of Geology and Mining of Stanford University, succeeding **Dr. John C. Branner**.

James C. Roberts, recently connected with the Bureau of Mines, has been appointed Professor of Safety and Efficiency Engineering of the Colorado School of Mines, which department was established by the trustees as a memorial to the late **Dr. Joseph Austin Holmes**.

Ambrose N. Diehl has been appointed assistant general superintendent of the Duquesne steel works of the Carnegie Steel Co.

Homer L. Carr has accepted the position of instructor in mining at the School of Mines, Columbia University, New York, N. Y.

William D. Burcham has resigned his position as mill superintendent of the Presidio Mining Co., Shafter, Tex., to open up some mines in the Terlingua quicksilver district.

Luciano Selmi, formerly chief chemist of the Otis Steel Co., has accepted a position with the River Furnace Co., Cleveland, where he will have charge of the chemical and physical laboratories of the new steel plant.

James F. McCarthy has been appointed manager of the National Copper Mining Co., Mullan, Idaho.

Walter M. Dake, Jr., has accepted the position of mill superintendent of the Aurora Consolidated Mines Co., Aurora, Nev.

P. K. Lucke has been made consulting mining engineer to the Cia. Minera de Penoles, El Paso, Tex., and the Cia. de Minerales y Metales, Monterey, Mexico.

C. G. Schlereth has been appointed assistant general superintendent of the Cia. Minera de Penoles, Mapini, Durango, Mexico.

Remember the New York Meeting, Feb. 14 to 17, 1916.

Robert Marsh, Jr., has left the Braden Copper Co. as consulting superintendent of mines, and has been appointed general superintendent of mines for the Nevada Consolidated Copper Co., Ruth, Nev.

Alexander Grosberg, general superintendent of mills of the Patino Tin Mines, Uncia, Bolivia is visiting the United States, his address being 4870 Fountain Ave., St. Louis, Mo.

Gerald G. Dobbs, formerly chief engineer of the ore mines and quarries in the southern district of the Republic Iron & Steel Co., has been appointed superintendent of the Raimund mining division for the same company at Bessemer, Ala.

J. D. Mooney, formerly in charge of the belting sales of the B. F. Goodrich Company, Akron, Ohio, is now general sales manager of the Hyatt Roller Bearing Company, Newark, N. J.

William J. Wolf, has resigned the position of resident manager for the Four States Coal & Coke Co., Worthington, W. Va., to become superintendent of the Carolina and Idanay mines of the Consolidation Coal Co., Fairmont, W. Va.

Gilmore Goodland. There is held at this office considerable mail matter addressed to Mr. Goodland from London, England, and we would be pleased to get in touch with him and ascertain what to do with this mail.

POSITIONS VACANT

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons.)

Assayer and chemist with experience in gold and silver work and refining; used to handling a force of men. Large property in Northern Ontario. Send reference, record of practice, age, married or single, and salary expected. Apply Position 54.

Metallurgical chemist wanted, competent to make the following analyses: cupola gray iron; air-furnace iron; pig iron of various kinds; coal; coke; sand of all descriptions; materials used in modern foundry practice; steel, and bronzes of all kinds; oils for fuel in Diesel engines and lubricating purposes; materials used in a modern machine shop, in fact, all materials likely to be used in a foundry, machine shop and forge shop. No. 55.

ENGINEERS AVAILABLE

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons introduced by members.)

Member, technical graduate, aged 36. Experienced copper metallurgist and mining geologist. Familiar also, through 15 years' broad practical experience, with mining, cyanidation, concentration (including oil flotation), metallurgical accounting, and mine and plant management. Best references. No. 237.

Member, graduate engineer, aged 44, with over 22 years' experience in connection with the development of large coal and coke properties in different parts of the country, desires position as general manager or general superintendent of large bituminous coal producing, or will take an interest and the management of a smaller operation. No. 258.

Member, aged 37, married, 15 years' experience as superintendent and general superintendent of coal and coking plants, open for engagement. Capable, a hustler, can handle labor and produce satisfactory results where possible. Closest investigation of ability invited. No. 264.

Member, technical graduate, B. S. and E. M., aged 32, married, nine years' varied experience as assayer, millman, metallurgist, chemist and testing engineer on flotation. Two years' teaching in West. At present in private work. Open for engagement after Dec. 1, 1915. Best of references. No. 265.

Superintendent and engineer, aged 38; technical graduate, exacting school; 14 years' experience in mining, 12 with present company, a large copper producer of Southwest; thorough miner; excellent engineer and geologist; familiar with Southwestern practice in milling and smelting; a systematizer and can organize the human element for results; of unquestioned reputation in district; seeks new connection on account of shutdown; exceptional references. No. 266.

Member, mine or mill superintendent, open for engagement; 12 years, experience in mining and ore dressing; experienced in mines with wide and narrow veins. Up to date in cyaniding, concentration, flotation. Fluent Spanish. Highest testimonials. Moderate salary. No. 267.

Member open for engagement as coal mine safety inspector; 24 years, experience in all classes and kinds of mines. Will go anywhere in United States or Canada. No. 268.

Experienced superintendent of metal mines and mills, aged 43, has specialized in cyaniding. Employed at present, desires change. Efficient organizer. Wishes to locate in East or Southeast, near good school facilities, with a chance to acquire interest in property needing better handling to make a success. High class references from former employers. No. 269.

Member, aged 29, technical education; seven years' experience in all phases of practical mine operations. Aerial tram construction and operation. Safety organization work, etc.; five years as mine superintendent. Available Jan. 1, 1916, as assistant to consulting engineer, or superintendent of small mine. United States preferred. No. 270.

Member, aged 32, married, college graduate, with two years' railway experience and six years' developing and operating large coal mining properties, desires change. At present chief engineer and assistant general superintendent of one of the largest producing companies of the Middle West. No. 271.

MONTANA LOCAL SECTION*Executive Committee***F. M. SMITH, Chairman****J. L. BRUCE, Vice-Chairman****D. C. BARD, Secretary, Box 267, Butte, Mont.****FREDERICK LAIST****W. C. SIDERFIN**

The semi-annual meeting of the Montana Section was held at the Silver Bow Block, Butte, Mont., on Friday evening, Oct. 8, 1915. An informal, subscription dinner, at \$1.50 a plate, was enjoyed in the early evening, by 62 members and guests.

After dinner a technical session was held, and a paper entitled Preliminary Outline of Paper on Mines in the Boulder Batholith was presented by J. A. Grimes, who is joint author of the paper, with Paul Billingsley. The paper elicited a lively and interesting discussion, in which a score of members took part.

After the reading of the paper the rest of the evening was spent socially.

D. C. BARD, Secretary.

AFFILIATED STUDENT SOCIETY NOTES

The Mining Engineering Society of the State College of Washington, Pullman, Wash., has elected the following officers for this semester:

F. MASON, President,**WILLIAM WHITE, Vice-President,****L. E. TRAEGER, Secretary and Treasurer.**

The Society holds meetings every two weeks at which papers on mining and metallurgical topics are presented by students and members of the faculty. At the meeting of Oct. 25, Prof. H. R. Robbins gave an interesting talk upon the Rankin Process for the treatment of base-metal ores. In this process the metals are extracted by the use of nitric acid, the acid being recovered almost entirely. Prof. Robbins thinks the process has great possibilities, but it has not yet been worked on a commercial scale.

The Missouri Mining Association, the student organization at the Missouri School of Mines, Rolla, Mo., has elected the following officers for the ensuing year:

L. W. EHLERS, President,**H. A. KLUGE, Treasurer,****J. G. REILLY, Vice-President,****N. E. RITTER, Secretary.**

Durward Copeland of the Missouri School of Mines has given a series of lectures relative to his trip through South America, and Lewis A. Delano recently gave a talk on Ore Dressing and Flotation at Flat River, Mo.

The Association is planning to have at least one lecture a month by some out-of-town man, and several talks by members of the faculty and students who have been in practice.

The University of Illinois Student Branch of the American Institute

of Mining Engineers, Urbana, Ill., has elected the following officers for the ensuing year:

H. E. WILLSON, <i>President</i> ,	R. R. KRAFT, <i>2nd Vice-President</i> ,
N. M. MORRIS, <i>1st Vice-President</i> ,	M. J. REED, <i>Secretary and Treasurer</i> .

The Mining Society of the University of Washington, Seattle, Wash., has elected the following officers for the ensuing year:

F. S. PORTER, <i>President</i> ,	H. G. BOULTON, <i>Secretary</i> ,
JESS C. JOHNSON, <i>Vice-President</i> ,	P. G. DOBSON, <i>Treasurer</i> ,
C. F. BLOGG, <i>Corresponding Secretary</i> .	

The School of Mines Society of the University of Minnesota has elected the following officers for the year:

ADOLPH DOVRE, <i>President</i> ,	OSCAR LEE, <i>Vice-President</i> ,
EDWIN SWEETMAN, <i>Secretary and Treasurer</i> .	

On Sept. 27, G. J. Young gave an illustrated talk on certain of the European mining districts which he had visited.

On Oct. 9, the annual reception for the freshmen of the School of Mines was held under the auspices of this Society.

The Society has a room in the new School of Mines which is used as a general rest and smoking room by the members.

The Chemical Society of Tufts College, Medford, Mass., has elected the following officers for the ensuing year:

CLAYTON R. SPENCER, <i>President</i> ,	ARTHUR DONNELLAN, <i>Treasurer</i> ,
MADELINE BIXBY, <i>Vice-President</i> ,	LEO T. MURPHY, <i>Secretary</i> .

Executive Committee

L. N. TOWNE,	C. R. SPENCER,	D. DEMPSEY.
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The Mining Society of Pennsylvania State College, State College, Pa., has elected the following officers for the fall term:

G. C. FAUST, <i>President</i> ,	R. C. BOYLES, <i>Treasurer</i> ,
C. F. FELEDY, <i>Vice-President</i> ,	M. E. GOETZ, <i>Secretary</i> .

Executive Council

J. C. HUB,	R. G. HARPER,	J. J. SUTTON,
C. M. NEVIN,	R. W. MILLER,	DR. E. S. MOORE.

During the last year the Society was addressed by J. H. Lewis, Pittsburgh, on The Coal Fields of British Columbia; C. A. Starr, Shamokin, Fighting Mine Fires; J. E. Johnson, Jr., New York, Power Plants for the Iron and Steel Industry; and A. G. McKee, Cleveland, Blast-Furnace Construction.

It is hoped to stimulate discussion at the meetings this season by having the students present papers on topics of interest, thereby also affording them opportunity for correct expression and promoting a feeling of cordiality among those who attend the meetings.

GUY C. FAUST, *President*.

Remember the New York Meeting, Feb. 14 to 17, 1916.

LIBRARY

AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS

AMERICAN SOCIETY OF MECHANICAL ENGINEERS

AMERICAN INSTITUTE OF MINING ENGINEERS

UNITED ENGINEERING SOCIETY

WILLIAM P. CUTTER, Librarian

The Library of the above-named Societies is open from 9 A.M. to 10 P.M. on all week-days, except holidays, from Sept. 1 to June 30, and from 9 A.M. to 6 P.M. during July and August. The Library contains about 55,000 volumes, including sets of technical periodicals and the publications of scientific and technical societies.

Members of the Institute, with few exceptions, are forced to spend a portion of their time in localities isolated from sources of information. To these the Library can render valuable service through correspondence: letters requesting information will receive special attention. The Library is prepared to furnish references and copies of articles on mining and metallurgical subjects; to determine the existence of mining maps, and to furnish general information as to the geology and mineral resources of all countries.

All communications should be made as definite as possible so that the information received may be what is desired and not include collateral matter which may not be of interest. The time spent in searching for such collateral matter will be saved, and the information will be sent more promptly and in more usable shape.

The members of the Institute can be of service to the Library by forwarding copies of mining reports, maps privately issued, and similar material, which will be classified, indexed, and made available to other members. Suggestions for additions to the Library, either by purchase or personal solicitation as gifts, will be welcomed. It is hoped that members while in the city will use the Library freely, and assurance is given that most careful service will be rendered to them.

LIBRARY ACCESSIONS**PARTIAL LIST CLASSIFIED BY SUBJECTS****Mining, Metallurgy and Chemistry**

PRACTICAL SURVEYING FOR SURVEYORS' ASSISTANTS, VOCATIONAL, AND HIGH SCHOOLS.
By Ernest McCullough, C. E. D. Van Nostrand Co., New York, 1915. Price \$2. (Gift of Publishers.)

[NOTE.—This manual contains in 400 pages a large amount of elementary instruction and practical advice for young surveyors. Even those who, having already received mathematical training, do not need that part of the book, will find in it a great deal which is not taught in the schools or set forth in the ordinary text-books. We refer particularly to the discussion of the legal duties and rights of surveyors in such matters as the restoration of lost boundaries, etc. The chapter in "Surveying Law and Practice" includes the whole of Chief Justice Cooley's famous address on the judicial functions of surveyors. As a manual for self-instruction this book, is admirably clear and well arranged, and it is surprising that a work intended primarily for that purpose should contain so much that makes it useful to advanced practitioners. It is, moreover, interesting and "readable," even to laymen.—R. W. R.]

MINING AND MINE VENTILATION. A Practical Handbook on the Physics and Chemistry of Mining and Mine Ventilation. By Joseph J. Walsh, Mine Inspector, Wilkes-Barre, Pa. D. Van Nostrand Co., New York, 1915. Price \$2. (Gift of Publishers.)

[NOTE.—This little book of 180 pages is intended for use in vocational schools, and for students qualifying themselves to be mine foremen and inspectors. It contains elementary instruction on matter, motion, force, gravitation, liquids, heat, gases, etc., as preliminary to practical discussions of the conditions, methods and apparatus of mine ventilation—chiefly in collieries. Each chapter is followed by questions to be answered and problems to be solved; and numerous tables, containing useful data, are scattered through the book. The title is misleading, since no other part of mining is treated than ventilation. For its limited purpose, the work is likely to be useful and convenient.]

ANALYSIS OF NON-FERROUS ALLOYS. By Fred Ibbotson and Leslie Aitchison. London, 1915.

BIBLIOGRAPHY OF MINNESOTA MINING AND GEOLOGY. Bull. No. 4, Minnesota School of Mines Experiment Station. Minneapolis, 1915. (Gift of University of Minnesota.)

BLAST FURNACE AND THE MANUFACTURE OF PIG IRON. Ed. 3. By Robert Forsythe. New York, 1913.

THE CHEMISTRY OF COLLOIDS AND SOME TECHNICAL APPLICATIONS. By W. W. Taylor. New York, 1915.

Geology and Mineral Resources

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QUEBEC. GEOLOGY OF A PORTION OF FABRE TOWNSHIP, PONTIAC COUNTY, QUEBEC. Quebec, 1911.

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General

CONCRETE STEEL CONSTRUCTION. Part 1—Buildings. By H. T. Eddy and C. A. P. Turner. Minneapolis, 1914.

DISCOVERIES AND INVENTIONS OF THE TWENTIETH CENTURY. By Edward Cressy. New York, 1915.

ELECTRICITY IN GASES. By J. S. Townsend, Oxford, 1915.

ILLINOIS COAL ON SPECIFICATION, PURCHASE AND SALE. Bull. No. 29, Illinois State Geological Survey. Urbana, 1914.

MATERIALS OF CONSTRUCTION: THEIR MANUFACTURE, PROPERTIES, AND USES. By A. P. Mills. New York, 1915.

NOTES ON MILITARY EXPLOSIVES. Ed. 3. By E. M. Weaver. New York, 1915

PURCHASING, ITS ECONOMIC ASPECTS AND PROPER METHODS. By H. B. Twyford. New York, 1915.

WORKING DATA FOR IRRIGATION ENGINEERS. By E. A. Moritz. New York, 1915.

Company Reports

- BROKEN HILL PROPRIETARY COMPANY, LTD.** Report of 80th half-yearly ordinary general meeting, Aug. 27, 1915. N. p. n. d. (Gift of Company.)
- ORIENTAL CONSOLIDATED MINING COMPANY.** Report for the year ending June 30, 1915. New York, 1915. (Gift of Company.)

Trade Catalogues

- BESSEMER GAS ENGINE Co.,** Grove City, Pa. Bessemer Monthly, Oct., 1915.
- CAMERON STEAM PUMP WORKS,** New York, N. Y. Bull. 154. Cameron Centrifugal pumps. June, 1915.
- DIAMOND POWER SPECIALTY Co.,** Detroit, Mich. Consulting engineers' guide. Specifications covering various types of "Diamond" Mechanical Soot Blowers.
- DURIRON CASTINGS Co.,** New York, N. Y. The story of "Duriron," as told in a series of advertisements. 1915.
- GENERAL ELECTRIC Co.,** Schenectady, N. Y.
Bull. No. 44,005. Line material and rail bonds for mine and industrial haulage. Sept, 1915.
- Bull. No. 44,012. Electrical equipment for the Puget Sound lines of the Chicago, Milwaukee and St. Paul Railway. Oct., 1915.
- Bull. No. 47,010. Standard unit direct-current switchboard panels for railway service. Aug., 1915.
- NATIONAL TRANSIT Co.,** Oil City, Pa. Bull. 403. Horizontal gas engines.
- OLIVER CONTINUOUS FILTER Co.,** San Francisco, Cal. Oil flotation—Oliver filtration. 1915.
- POMONA MANUFACTURING Co.,** Pomona, Cal. Deep well pumps, circular.
- WEBSTER MFG. Co.,** Tiffin, O. Webster method. Sept., 1915.

Remember the New York Meeting, Feb. 14 to 17, 1916.

MEMBERSHIP

NEW MEMBERS

The following list comprises the names of those persons who became members during the period Oct. 10 to Nov. 10, 1915:

Members

- BECK, WESLEY HUDSON, Civ. and Min. Engr., Ellsworth Collieries Co., Ellsworth, Pa.
 BRADSHAW, GRANT D., Pres., Andrews-Bradshaw Co,
 409 Diamond Bank Bldg., Pittsburgh, Pa.
 BUTLER, BERT S., Geol. U. S. Geological Survey, Washington, D. C.
 CAMERON, FRANK, Asst. Mine Foreman, Braden Copper Co.,
 Rancagua, Chile, So. Amer.
 CHEN, YEFAN FRANCIS, Asst. Min. Engr., Pinghsiang Colliery,
 Pinghsiang-Ki, via Changaha, China.
 CLAGHORN, DAVID MONTGOMERY, Mining Engr., Timekeeper,
 Wilkeson Coal & Coke Co., Wilkeson, Wash.
 DAVIS, REUBEN WALDO, Min. Engr., Supt., Cannel Coal Co., Dolores, Webb Co., Tex.
 DOUTHETT, GEORGE M., Supt. Crescent Metals Co., Glenwillard, Pa.
 DUNSFORD, ENSOR R., Min. Engr., Asst. Supt., Victoria Gypsum Min. & Mfg. Co., Ltd.,
 Quarry, St. Anns, C. B., Nova Scotia, Canada.
 FORBES, FREDERICK B., Min. Engr., Care F. A. Pellas & Co., Granada, Nicaragua,
 Central America.
 GEPP, HERBERT WILLIAM. 62 Gates Ave., Montclair, N. J.
 GILLETTE, G. MARSHALL, Mgr. The Consolidation Coal Co., Jenkins, Ky.
 HALL, OLIVER, Mines Supt. Mond Nickel Co., Coniston, Ont., Canada.
 HAND, ORVILLE M., Supt. Queen of the West Mines Co., Cornucopia, Ore.
 HENEGAR, HERBERT BENTON, Mine Foreman, Care American Zinc Co., Mascot, Tenn.
 HOLIDAY, FRANCIS AUGUSTUS, Geol., Institute of Petroleum Technologists,
 17 Grace-church St., London, E. C., England.
 KNAPENBERGER, WILLIAM RALPH, Mill Asst. . . American Zinc Co., Mascot, Tenn.
 LASKY, BERNARD HAROLD, Min. Engr. Kennett, Shasta Co., Cal.
 LEONARD, WILLIAM DIXON, Supt., American Smelt. & Ref. Co.,
 Garfield Smelter, Garfield, Utah.
 LEWIS, CLAUDE FREEMAN, Div. Supt., Philadelphia & Reading Coal & Iron Co.,
 Pottsville, Pa.
 LINDEN, BERNARD HUGO VAN DER, Chief Geol. and Field Mgr.,
 Shell Company of Cal., San Francisco, Cal.
 LINDSAY, BRUCE R., Mine Supt., Republic Iron and Steel Co.,
 Martin, Fayette Co., Pa.
 LINDSEY, HARLEY MARTIN, Shift Boss. Ray Hercules Copper Co., Ray, Ariz.
 MCFADDEN, GEORGE CALDWELL, Min. Engr. Lock Box 560, Witt, Ill.
 MACDONALD, HENRY DOUGLAS, Met., Chief Sampler, American Zinc Co.,
 Mascot, Tenn.
 MACREADY, GEORGE ALEXANDER, Resident Geol., Trinidad Lake Petroleum Co., Ltd.,
 Brighton, Trinidad, B. W. I.
 MARRIOTT, ALEXANDER DODSLEY, Jr., Engr., Dort Cyanide Mach. Co.,
 Box 6, P. P. I. E., San Francisco, Cal.
 MITCHELL, LEROY BRADLEY, Min. Engr., Copper Queen Cons. Min. Co., Bisbee, Ariz.
 NANCE, W. FRED, Mine Supt. Belknap Coal Co., Newcastle, Tex.
 PATTERSON, ROBERT DUN, Mine Mgt., Weyanoke Coal & Coke Co.,
 Lowe, Mercer Co., W. Va.
 PERRY, ROBERT MATSON, Min. Engr., Genl. Supt., The Moffat Coal Co.,
 Oak Creek, Colo.
 RALSTON, OLIVER C., Asst. Met., U. S. Bureau of Mines,
 Care University of Utah, Salt Lake City, Utah.
 ROGER, EUGENE, Engr., Union Miniere du Haut Katanga,
 Elisabethville (Katanga) Congo Belge, via Capetown.
 SMITHER, THOMAS M., Min. Engr. Nevada Wonder Mining Co., Wonder, Nev.
 WILLIAMS, WILLIAM HENRY, Vice-Pres., The Delaware and Hudson Co.,
 32 Nassau St., New York, N. Y.

Associate Members

BURNS, JAMES FERGUSON, Pres., Cripple Creek Deep Drainage and Transportation Tunnel Co., Colorado Springs, Colo.
 BRADLEY, WILLIAM M., Counsellor.....405 Felt Bldg., Salt Lake City, Utah.
 THOMAS, PHILIP EDWARD, Coal Mining.....1327 Spruce St., Philadelphia, Pa.

Junior Members

KLINE, ALLEN HAROLD.....46 Fairview Ave., Naugatuck, Conn.
 KONSELMAN, ALBERT S., Min. Engr.....Care Engineering Dept., 4 C, Naco, Ariz.
 ZIMMER, ERNEST BARNARD, Chemist, Alliance Works, American Steel Foundries, Alliance, Ohio.

Reinstated Members

HAMILTON, THOMAS M., Min. Engr.....Chanaral, Chile, So. Amer.
 WALLACE, LOUIS R., Supt.....International Smelting Co., Miami, Ariz.

Total Membership, Nov. 10, 1915 5,305

CANDIDATES FOR MEMBERSHIP

APPLICATION FOR MEMBERSHIP.—The Institute desires to extend its privileges to every person to whom it can be of service. On the other hand, it is not desirable that persons should be admitted to membership in classes for which they are not qualified. Members of the Institute can be of great service if they will make a practice of glancing through the list of applicants and promptly notifying the Committee on Membership or the Secretary of the Institute, of any persons whom they think should not be classified in accordance with the list given.

The following persons have been proposed during the period Oct. 10 to Nov. 10, 1915, for election as members of the Institute. Their names are published for the information of Members and Associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Board of Directors, which has the power of final election.

Members

Maxwell Wanton Atwater, Basin, Mont.

Proposed by Robert Linton, Rene de Sallier du Pin, D. C. Bard.

Born 1878, Millville, N. J. 1886-90, Grammar and private schools, Millville, N. J. and Germantown, Pa. 1890-92, Realschule, Berlin, Germany. 1892-96, Preparatory School, Wm. Jenner, Syracuse, N. Y. 1896-97, University College, London. 1897-1901, Colorado School of Mines, Golden, Colo. E. M. 1901-03, Miner, Engr., Asst. to Supt., Liberty Bell G. M. Co., Telluride, Colo. 1904-05, Supt., United Elkhorn Mines, Baker Co., Ore. 1905-09, Jr. partner, Atwater, Linton & Atwater, examinations and management, Montana, Colorado, Bolivia, Mexico. 1909-12, Genl. Supt., Butte & Superior Copper Co., Ltd., Butte, Mont.

Present position: Owner and Manager, zinc concentrator and mining property.

William A. Beaudry, Kellogg, Idaho.

Proposed by Ernest G. Gnaedinger, Rush J. White, Fred W. Callaway.

Born 1878, Houghton, Mich. General school education. 1896-1900, Miner, Garden City Development Co., Homestake Min. Co., Black Hills, S. D. 1900-01, Miner, Republic, Wash.; Phoenix B. C. 1901-06, Miner, Timberman, Contractor, Last Chance, Bunker Hill, Black Hawk, etc., Wardner, Idaho. 1906-08, Foreman, Stewart Mine, Kellogg, Idaho.

Present position: 1908 to date; Supt., Stewart Mine and Coeur d'Alene Development Co.

Carlos Billick, Keddle, Cal.

Proposed by Thomas T. Read, H. L. Slosson, W. S. Morley.

Born 1889, Downieville, Cal. 1914, Univ. of California, B. Sc., Berkeley, Cal. 1914, Nevada Cons. Copper Co., McGill, Nev. 1914-15, Associated with I. W. Gibbons in general mining work, Alturas, Cal.

Present position: Oil Flotation Operator, Engels Copper Min. Co.

William J. von Borries, Joplin, Mo.

Proposed by C. A. Wright, H. A. Buehler, Herbert Taylor.

Born 1881, Louisville, Ky. 1889, Du Pont Manual Training School, Louisville, Ky. 1901-02, Rose Polytechnic Institute, Terre Haute, Ind. 1905, Lehigh Univ., So. Bethlehem, Pa., E. M. 1905-07, U. S. Geological Survey, Fuel Testing Plant, St. Louis. 1907-09, Supt. of Mines, Coalmont, Tenn. 1909-10, General engineering & construction practice, Ky. & Tenn. 1910-13, Genl. Mgr., lead and zinc mining properties, s.w. Mo. 1913-14, Mining eng. practice, Ky., Va., Tenn. & West.

Present position: Genl. Supt. of Mines.

Charles H. Briggs, Bonne Terre, Mo.

Proposed by C. J. Adami, Alan D. Bell, L. A. Delano.

Born 1883, Springfield, Mo. 1902, Central College, Fayette, Mo., A. B. 1905, Washington Univ., St. Louis, Mo., B. S. 1905-08, Construction work, Republic Iron and Steel Co., East St. Louis, Ill. 1908-09, Mill designing, Doe Run Lead Co., Rivermines, Mo.

Present position: 1909 to date; Construction Engr., St. Joseph Lead Co.

L. C. Church, Joplin, Mo.

Proposed by A. F. Truex, H. A. Buehler, C. A. Wright.

Born 1877, Cincinnati, O. 1896-97, Mass. Inst. of Tech. 1897-99, Stanford Univ., Mining Dept. 1903-04, Columbia Univ. School of Mines.

Present position: Have been operating my own zinc and lead properties in the Joplin district for the past eleven years.

Lloyd J. Davis, Joplin, Mo.

Proposed by H. S. Young, F. C. Wallower, Charles T. Orr.

Born 1876, Burlington, Vt. To 1893, Public Schools, Fitchburg, Mass. 1897, Grad., Worcester Polytechnic Institute, B. S. 1898-99, United Shoe Machinery Co. 1899-1903, Sullivan Machinery Co. 1903-04, Robinson Machinery Co.

Present position: 1905 to date; Sales Engr., Ingersoll Rand Drill Co.

Sidney Hugh Davis, Joplin, Mo.

Proposed by C. A. Wright, Philip N. Moore, Herbert DeQ. Taylor.

Born 1886, Morristown, N. Y. 1905-09, Univ. of Wisconsin, B. S. 1909-10, Supt. of zinc mine, Vinegar Hill Zinc Co., Wisconsin Zinc and Lead District.

Present position: 1911 to date; Mgr., zinc mines, Vinegar Hill Zinc Co. and Century Zinc Co., Joplin District.

John Howard Dugan, Hazleton, Pa.

Proposed by Douglas Bunting, R. V. Norris, Walter Fahringer.

Born 1860, Lairdsville, Pa. 1866-77, Public School. 1877-79, Grad., West Chester State Normal School. 1879-82, Taught School and worked on Eng. Corps, studying. 1883-91, Mining Eng., Lehigh and Wilkes-Barre Coal Co. 1891-1901, Contractor, Lehigh and Wilkes-Barre Coal Co., and Lehigh Valley Coal Co., and Cox Bros. Co. 1901-03, Genl. Mgr., Chinese Eng. & Min. Co., Ltd., Tongshan, China. 1903-06, Supt., Dodson Coal Co., Morea Colliery, Pa.

Present position: 1906 to date; Stripping Contractor, Lehigh and Wilkes-Barre Coal Co. and C. M. Dodson & Co.

Lindsay Duncan, McGill, Nev.

Proposed by C. B. Lakenan, Charles F. Rand, George A. Howells, W. L. Saunders, Pope Yeatman, R. E. H. Pomeroy.

Born 1875, Brooklyn, N. Y. 1892-97, Univ. of Maine. 1897-99, Clark Univ. 1902-03, Cornell Univ., B. S., M. S. 1899-1901, Instructor, Union University. 1901-02, Supt. of Const., Amer. Locomotive Co., Schenectady, N. Y. 1902-03, Instructor, Cornell Univ., Ithaca, N. Y. 1903-04, Prof. of C. E., Univ. of Colo. 1904-06, Engr., Gunnison Tunnel, Montrose, Colo. 1905-07, Engr., A. S. & R. Co., Denver, Colo.

Present position: 1907 to date; Mech. Engr., Nevada Cons. Copper Co.

Charles Ferry, Bridgeport, Conn.

Proposed by Arthur F. Taggart, J. F. McClelland, L. W. Bahney.

Born 1868, Peterboro, N. H. 1888, Worcester Polytechnic Institute, B. S. 1880-90, Taught in Inorganic Chem. Analysis at the Mass. Inst. of Tech.

Present position: 1895 to date; Bridgeport Brass Co.

Thomas E. Fisher, New York, N. Y.

Proposed by Chester A. Fulton, L. O. Kellogg, Louis D. Huntoon.

Born 1883, Lansingburg, N. Y. 1897-1901, Mt. Vernon High School. 1901-05, Columbia Univ., E. M. 1905-07, Engr., Guanajuato Cons. M. & M. Co., Guanajuato, Mex. 1907-08, Chem., Dwight Furnace Co., Guanajuato, Mex. 1908-09, Engr., Guanajuato Development Co. 1909-10, Engr., Guanajuato Amalgamated Co. 1910-11, Mech. Engrg. Dept., Cia. Real de Monte. 1911-12, Asst. Mgr., Aguacate Mines, Costa Rica.

Present position: 1912 to date; Engr., Underpinning Foundation Co.

James Russel Fleming, Urbana, Ill.

Proposed by H. I. Smith, F. W. DeWolf, H. H. Stoek.

Born 1886, Arnot, Tioga Co., Pa. Common School. 1904-07, Grove City College, Grove City, Pa. 1908-11, School of Mines, Univ. of Pittsburgh, Pittsburgh, Pa., E. M. 1908-10, Engineering corps and various mining work. 1911, Mining class work, Bit. Comm., Y. M. C. A. 1911-12, Min. Engr., Day & Zimmerman, Philadelphia, Pa., on central power plant work, Central Penn. district. 1912 to date; U. S. Bureau of Mines.

Present position: Asst. Min. Engr., U. S. Bureau of Mines.

Peter Andreas Haines, Rivermines, Mo.

Proposed by Ambrose E. Ring, E. J. Rossbach, C. J. Adami.

Born 1876, Lofoten, Norway. 1899-1901, Elementary School of Mines, Kongsberg, Norway. 1906-09, Montana State School of Mines, M. E. 1902, Supt., Huglin Copper Mine, Huglin, pr. Stordo, Norway. 1909-10, Testing Engr., Washoe Reduction Wks., Anaconda, Mont. 1910-11, Supt., Del Rio Min. Co., Elk City, Idaho.

Present position: Div. Supt., Doe Run Lead Co.

P. R. Hines, El Paso, Tex.

Proposed by David Cole, Julius Bergman, W. S. Harrison.

Born 1886, Toledo, O. 1901-05, New Mexico School of Mines, Socorro, N. M. 1905-07, Michigan College of Mines, Houghton, Mich., E. M. 1905, Miner, Cripple Creek. 1907-08, Engr., St. Anthony Reef, Sturgeon Lake, Ont. 1908, Surveyor's Helper, Federal M. & S. Co., Wallace, Idaho. 1908-10, Engr., Mill Foreman, Minas Pedrazzini, and Supt., El Carmen Mine, Arizpe, Sonora. 1910-11, Millman, Test Man, various companies Arizona. 1912, Draftsman, David Cole, Morenci, Ariz. 1913-15, Construction Engr., in charge of Flotation Dept., Caucasus Copper Co., Ltd., Batum, Russia. 1915, Concent. Draftsman, Inspiration Cons. Co., Miami, Ariz.

Present position: In charge of drafting and design new concentrator, San Francisco Mines of Mexico.

David Leavitt Hough, Petrograd, Russia.

Proposed by W. L. Saunders, George A. Howells, George T. Cousins.

Born 1865, Fort Wayne, Ind. 1885, Yale Univ., Ph. B. 1885-87, Apprentice and Journeyman Machinist. 1887-92, Draftsman to Chief Engr., R. D. Wood & Co., Philadelphia. 1892-95, Chief Engr., East River Gas Co., built tunnel under East River. 1895-98, Mgr., National Contracting Co., Barber Asphalts Interest. 1898-99, Capt., U. S. V. Engineers. 1899-1914, Contractor, particularly on tunnel work; P. R. R. tunnels across Manhattan, under East River, and Harbor of Havana.

Present position: Senior member Technical (Contracting Engineering) and Trading (Engineering Equipment) Co., "Hough-Kolensky Co."

Ichiro Ishikawa, New York, N. Y.

Proposed by Edward L. Young, Shigema Yamanouchi, Takeshi Kawamura.

Born 1885, Tokyo, Japan. 1906, Grad., First Higher School (preliminary course to the University). 1909, Grad., Tokyo Imperial Univ., Dept. of Applied Chem.

Present position: Asst. Prof., Tokyo Imperial Univ., Dept. of Applied Chem. and Cons. Chem., Kanto Acid and Alkali Co.

Fred G. Jarrett, Carbonado, Wash.

Proposed by Percy E. Wright, Joseph Daniels, Milnor Roberts.

Born 1874, Forest of Dean, England. 1890, Monmouth Grammar School, England. 1896, Certificate Mining School, Newcastle of S. Wales, Aust. 1898, Manager's Certificate, Aust. 1909, Mining course, International Correspondence School. 1910, Foreman's Certificate, B. C. 1898, Deputy, undermanager and manager, Great Western Colliery, Vic., Aust. 1901, Shiftboss and undermanager, Malcom Star Gold Min. Co., W. Aust. 1902, Webster's Find Gold Min. Co., W. Aust. 1904, Govt. Exploration as prospector; Manager prospecting outfit, Fiji Legislative Council. 1912, Mine Foreman, Canadian Collieries, Vancouver Island, B. C.

Present position: Mine Foreman, Carbon Hill Coal Co.

Robert E. Jennings, 2nd, New York, N. Y.

Proposed by Albert Sauveur, H. M. Boylston, Bradley Stoughton.

Born 1884, Washington, D. C. Public School, Washington, D. C. Bradstreets preparatory school, Rochester, N. Y. 1904-07, West Point. 1910-11, Harvard School of Applied Science. 1907-10, Taylor Iron and Steel Co., High Bridge, N. J. 1911-13, The Titan Steel Casting Co., Newark, N. J. 1913-15, Genl. Cons. Engrg. Office, New York.

Present position: Hewitt Steel Corporation.

Chisato Kawamura, Sagaken, Japan.

Proposed by Edward L. Young, Shigema Yamanouchi, Takeshi Kawamura.

Born 1873, Tokyo, Japan. 1903, Grad., Tokyo Univ. of Japan. 1903, Chief Engr., Electric Light and Power Co., Shiguokaken, Japan. 1908, Engr., Kano Copper Mine, Fukushima, Japan.

Present position: 1911 to date; Chief Engr., Ochi and Yoshitani Coal Mine, Mitsubishi Co., Japan.

Samuel M. Keiper, New York, N. Y.

Proposed by Charles F. Rand, George W. Pfeiffer, Paul S. King.

Born 1844, near Middletown, Pa. Education received in public schools and White Hall Academy. Experience mostly in mining of iron ores in Pennsylvania and New Jersey. Also in the manufacture of explosives under the name of New York Powder Co. of which I was President and principal owner.

Present position: Retired.

Carl August Lantz, Pachuca, Mex.

Proposed by Hugh Rose, William J. Cox, Richard A. Parker.

Born 1883, Green Bay, Wis. To 1899, Common Schools, Milwaukee, Wis. and Colton, Cal. 1903, Grad., Redlands, Cal. High School. 1904-07, Chemistry Course, Leland Stanford Jr. Univ. 1907-08, Shiftboss, Cyanide Mill, Reduction & Mines Co., Guanajuato, Gto., Mex. Shiftboss, Cyanide Mill, Pinguico Mines Co., Guanajuato, Gto., Mex. 1908-09, Mill Supt., Cyanide Mill, Guanajuato Amalgamated Gold Mines Co., La Luz, Guanajuato, Mex. 1909, Mill Supt., Cyanide Mill, Pinguico Mines Co., Guanajuato, Mex. 1910-14, Mill Supt., Cia. Beneficiadora de Pachuca, S. A.

Present position: 1914 to date; Genl. Supt., Cia. Beneficiadora de Pachuca, S. A., and Cia. de Santa Gertrudis, S. A.

John George Leyner, Denver, Colo.

Proposed by W. L. Saunders, L. D. Albin, George A. Howells.

Born 1860, Boulder, Colo. 1870-80, District Schools. Prest., J. George Leyner Engrg. Works Co., Littleton, Colo. Inventor of Leyner drill. Inventor of Leyner drill sharpener. Was formerly engaged on machine shop practice repairing mining and contractors' machinery. Has had practical experience in lead and silver mining in Colorado.

Present position: Cons. Engr., Ingersoll Rand Co.

Kuo Ching Li, New York, N. Y.

Proposed by Thomas T. Read, T. A. Rickard, E. P. Jennings.

Born 1890, Changsha, China. 1903-08, Grad., Technical Institute, Hunan, China, M. E. 1912-15, Grad., Royal School of Mines, London, A. R. S. M. 1908-09, Secy., Hunan Mining Board. 1909-10, Mgr., Kuanwah Mine Co. 1910-11, Mgr., Sukaushan Lead & Zinc Mines.

Present position: Mining Commissioner, Min. Engr., Hunan Mining Board and Wah-Chang Min. & Smelt. Co.

Frank Austin Lidbury, Niagara Falls, N. Y.

Proposed by Joseph W. Richards, E. F. Roeber, Arthur L. Walker.

Born 1879, Middlewich, England. 1895-98, B. Sc.; 1900 M. Sc., Univ. of Manchester. 1898-1903, Research work, Physical chemistry, Manchester, London and Leipzig. 1903-05, Chief Chemist, Oldbury Electro Chemical Co. 1914-15, Pres., American Electro Chemical Society.

Present position: 1906 to date; Works Mgr., Oldbury Electro Chemical Co.

William Douglas McMillan, Schumacher, Ont., Canada.

Proposed by E. A. Hersam, W. S. Morley, E. B. Durham.

Born 1892, Muskegon, Mich. 1905-09, Eureka High School, Cal. 1909-14, Univ. of California, Berkeley, Cal., B. S. 1911, Assay office and with surveying crew, Alaska-Treadwell Mines. 1912, Mucker, North Star Mines, Cal. 1914, Field Asst. on Reconnoissance work, U. S. Geological Survey, Yosemite National Park.

Present position: 1914 to date; Asst. Engr. & Geo., McIntyre-Porcupine Mines, Ltd.

John D. MacKenzie, Boston, Mass.

Proposed by Waldemar Lindgren, H. O. Hofman, Edward E. Bugbee.

Born 1888, Baddeck, Cape Breton, Canada. 1911, Mass. Inst. of Tech., S. B.

Present position: 1912 to date; Instructor, Mass. Inst. of Tech., and Asst. Geol., Geological Survey of Canada.

Albert T. March, Collinsville, Ill.

Proposed by Jesse O. Betterton, W. E. Newnam, Herman Garlich.

Born 1885, Ocheyedan, Iowa. 1910, Univ. of Michigan, Bachelor of Chemical Engineering. 1910-12, Chemist, Omaha laboratories, Union Pacific R. R. Co. 1912-13, Night foreman, Bismuth and Experimental Plants. 1913-15, Night foreman Lead Refinery, Am. Smeltg. & Refg. Co.

Present position: Supt., River Smelt. & Ref. Co., Keokuk, Iowa.

Itsumaru Miyasaki, Fukuoka, Japan.

Proposed by Edward L. Young, Shigema Yamanouchi, Takeshi Kawamura.

Born 1875, Tokyo, Japan. 1902, Grad., Tokyo Univ., Japan. 1902, Chief Engr., Namaguta Coal Mine, Mitsubishi Co., Tokyo, Japan. 1906, Chief Engr., Shinnew Coal Mine, Mitsubishi Co., Tokyo, Japan.

Present position: 1912 to date; Chief Engr., Shinnew and Kanada Coal Mine, Mitsubishi Co.

Laurence Stillman Morrison, Grass Valley, Cal.

Proposed by J. B. Stapler, Thomas V. Reeves, Robert H. Bedford.

Born 1888, Redlands, Cal. 1902-05, Thacher School, Nordhoff, Cal. 1905-07, Phillips Academy, Andover, Mass. 1907-11, Yale College, B. A. 1911-12, Savings Bank of Redlands, Redlands, Cal. 1912-14, Wells Fargo Nevada Nat. Bank, San Francisco, Cal. 1914, Sampler, Surveyor, Specimen Boss, Empire Mines, Grass Valley, Cal.

Present position: Office Assistant.

Hajime Nakamura, Tochigi-ken, Japan.

Proposed by W. L. Saunders, George A. Howells, L. D. Albin.

Born 1879, Tokyo, Japan. 1900, Grad., Tokyo First High School. 1903, Grad., Civil Engineering Dept., Kyota Imperial Univ., Kogakushi. 1903, Const. Engr., Ashio Copper Mines, Furukawa Mining Co.

Present position: Chief of Const. Dept., Ashio Copper Mines.

Wilbur Armistead Nelson, Nashville, Tenn.

Proposed by A. H. Purdue, L. C. Glenn, S. W. McCallie.

Born 1889, Nashville, Tenn. 1910, Vanderbilt Univ., B. S. 1908-10, Instructor Chemical Lab., Vanderbilt Univ. 1915, Stanford Univ., M. A. 1910, Chem. Tennessee Packing Co., Nashville, Tenn. 1910-14, Asst. Geol., Tennessee Geological Survey, Nashville, Tenn. 1912 (summer), Asst. Geol., Virginia Geological Survey.

Present position: 1914 to date; Economic Geol., Nashville, Chattanooga and Saint Louis Ry.

Arvid E. Nissen, Gary, Ind.

Proposed by Harry E. Nold, George J. Young, W. R. Appleby.

Born 1890, La Crosse, Wis. 1913, E. M. 1914, Univ., of Minn., M. Sc. 1910, Surveyor, E. J. Longyear & Co. 1912, Miner, Calumet and Arizona Min. Co., Bisbee, Ariz. 1913, Appraisal Engr. of public utilities with Charles L. Pillsbury, St. Paul. 1914, Steel Inspector, Pittsburg Testing Laboratories, Minneapolis, Minn. Branch. 1915, Metallographer, Inland Steel Co.

Present position: Mixer Foreman, No. 1 Open Hearth, Illinois Steel Co.

P. J. O'Gara, Salt Lake City, Utah.

Proposed by W. H. Howard, P. A. Mosman, C. W. Whitley.

Born 1872, Coleridge, Nebr. 1895-96, Univ. of So. Dakota. 1898-1902, Univ. of Nebraska, B. Sc. 1904-05, Abroad, pursuing pathological studies. 1902-10, Pathologist, U. S. Dept. of Agriculture. 1910-14, Pathologist and Entomologist and Special Meteorological Observer, U. S. Weather Bureau, Medford, Ore.

Present position: Chief in charge of Agricultural Investigations, American Smelt. & Ref. Co.

Frederick Warren Osborn, Ajo, via Gila, Bend, Ariz.

Proposed by Charles E. Locke, Edward E. Bugbee, H. O. Hofman.

Born 1887, Edgartown, Mass. 1903-06, Worcester Academy, Worcester, Mass. 1906-14, Grad., Mass. Inst. of Tech., S. B. 1909-10, U. S. Engr. Corps, Fortification Dept., Boston. 1911 (summer), Inspector diamond drill work, Stone and Webster Eng. Corp., Kellyville, N. H. 1912-13, Engr., Ajo Cons. Copper Co. 1913-14, Engr. helper, Chino Copper Co.

Present position: Engr., Ajo Cons. Copper Co.

Frank W. Parker, Rye Valley, Ore.

Proposed by L. A. Walker, Howard S. Lee, Frederick Lyon.

Born 1870, Penn Yan, N. Y. Common School. 1887-89, Washington Univ. 1890-98, Homestake Mines, White Oaks, N. M. 1898-1904, Candelaria Mines, San Pedro, Mex. 1904-06, Various mines, Nevada and Idaho. 1906-10, Leasing in Idaho.

Present position: 1911 to date; Mine Supt., Rainbow Mine.

George Proud, Valparaiso, Chile.

Proposed by Mark R. Lamb, C. Garcia Cross, Walter Renton Ingalls.

Born 1870, Gateshead-on-Tyne, England. 1880-85, Dr. Rutherford's Academy, Gateshead, England. 1885-90, Engr. apprentice and student, Clarke-Chapman & Co., Gateshead. 1890-1903, Service at sea; eleven years as Chief Engr., latter five as Chief Engr. on Chilean Navy. Holder of Board of Trade 1st class certificate. 1903-15, Principal works in which I was engaged. Erection of several nitrate plants, superintendence of erection of Catemu y Melor Copper Smeltg. Works. Cons. Engr., to Société des Mines de Cuivre de Catemu (French Co.) and to several Nitrate Companies. Designing and superintendence of new water works for Valparaiso executed by contract by my firm.

Present position: Partner, Beaver-Proud Engineering Co.

Adelbert Harry Richards, Murray, Utah.

Proposed by C. W. Whitley, C. W. Adams, Jr., J. B. McIntosh.

Born 1878, Pueblo, Colo. 1897, Grad., Central High School with advanced credits in chemistry. 1898-99, Engrg. & Chemistry, Univ. of Colorado. 1900, Seranton Correspondence Metal Mining. For three years after this and during my work I studied chemical and engineering books and literature in a definite and consistent manner. 1899-1902, Asst. Chief Chemist, American Smelt. & Ref. Co., Philadelphia Plant, Pueblo, Colo. 1902-04, Chief Chemist and Assayer, Asst. Supt., Bromination and chlorination, General Metals Co., Colorado Springs, Colo. 1904-05, Chief Chemist and Assayer, chlorination work, Economic mill, Victor, Colo. 1905, Chemist, Copper Queen Smelter, Douglas, Ariz.

Present position: Asst. Supt., Murray Plant, American Smelting & Refining Co.

Edouard G. Saladin, New York, N. Y.

Proposed by Henry M. Howe, Arthur G. Levy, Albert Sauveur.

Born 1856, Nancy, France. 1875-77, Ecole Polytechnic, Paris. 1877, Licencié et Sciences Physiques. 1877-80, Ecole Supérieure de Mines de Paris, with Diploma of Civ. Min. Engr. 1887-88, Central laboratory, Société Internationale de Electricien, Paris. 1880-84, Geological surveys in France; mining explorations in Cochinchina, Tonkin, Greece, Bosnia. 1884-86, Copper mines of Vignobes, Norway. 1886-87, Copper mines of Company, Boleo, Mex. 1888-91, Sauter Lemounier works, Paris. 1891-94, Mining explorations for several banks. 1896-98, Supt. of works, Compagnie des placers Hydrauliques de Junction City, Trinity Co., Cal. 1898-14, Engr. (principal) of Schneider & Co.

Present position: Engr., French Artillery Commission.

William Noble Smith, Platteville, Wis.

Proposed by C. A. Wright, H. A. Buehler, Charles Page Perin.

Born 1878, Cresco, Iowa. 1897, Univ. of Wisconsin, B. A. Later post-graduate work in geology and allied subjects. 1904, Asst. Geol., U. S. Geological Survey.

Present position: 1905 to date; Genl. Mgr., Vinegar Hill Zinc Co.

Eugene Austin Stephenson, Chicago, Ill.

Proposed by Albert D. Brokaw, James H. Hance, W. H. Emmons.

Born 1884, Franklin, Mich. 1905, Adrian College, Adrian, Mich., B. S. 1915, Univ. of Chicago, Ph. D.

Present position: Instructor in Geology, Univ. of Chicago.

Harry Dukehardt Sultz, Butte, Mont.

Proposed by B. H. Dunshee, George E. Moulthrop, A. A. Zentner.

Born 1885, Baltimore, Md. 1899, Grad., Butte Public Schools. 1899-1901, Butte High School. 1901-02, Preparatory course, Montana School of Mines. 1907-11, Montana School of Mines, E. M. 1909-12, General Mining, Butte. 1912-13, Sampler and Asst. Engr., Anaconda Copper Min. Co., Butte. 1913-14, Engr., Mont., Ill. Copper Min. Co., Cardwell, Mont. 1914-15, Sampler, Anaconda Copper Min. Co.

Present position: Asst. Engr., Anaconda Copper Min. Co.

Percy William Thompson, Fullerton, Cal.

Proposed by Andrew C. Lawson, M. L. Requa, Walter Stalder.

Born 1881, Ontario, Canada. 1912, Grad., Univ. of Cal., College of Mining, B. S. 1912-14, On geological staff, Standard Oil Co., Cal., Washington and Alaska.

Present position: Asst. Supt., for one of the Standard Oil Properties.

George H. Utter, Silver City, N. M.

Proposed by Thomas H. O'Brien, Arthur Notman, Charles A. Mitke.

Born 1854, Green Co., New York. Common school education in Conn. Took special courses with Prof. Shedd, a graduated chemist and mining engineer with whom I was associated for several years, he as an assayer, chemist and mining work during the early eighties in Santa Fe, N. M. 1881-85, Assaying, developing examination prospects, Assay Office, Santa Fe, N. M. 1884, Appointed Fiscal Agent of N. M. and made its first credit by the sale of 200,000 Capital Building Bonds. 1885, At the request of the Santa Fe R. R. sold all of the Co's. R. R. Aid Bonds. 1886, At the request of the town of Silver City, N. M., was granted a franchise and built the Silver City Water Works; also mining. 1887-93, Mining in charge of Water Works. 1889-1915, Purchased the Cleveland group. Developed and made a mine during the next 20 years, also other mining works, including examinations.

Present position: Owner and Manager, Carlisle and other groups of developed mines.

Frederick A. Voorhees, Pearce, Ariz.

Proposed by Abbot A. Hanks, E. H. Lealie, Edgar A. Collins.

Born 1881, Hightstown, N. J. 1896, Hightstown High School. 1897-1900, Peddie Institute. 1901-04, Time-keeper, J. W. Burson, Copper prospect, Wickensburg, Ariz. 1905-06, Cyanide Shift-boss, Dos Estrellas Min. Co., Tlalpuhagua, Mich., Mex. 1906-07, Assayer, Ajuchitlan M. & M. Co., Tejasquiapan, E. de Quiretoro, Mex. 1907-08, Smelter-Refinery and Zinc-house, Peregrina M. & M. Co., Guanajuato, Gto., Mex. 1909-10, Cyanide Supt., Loreto Mill, Cia. Real del Monte, Pachuca, Hgo. 1911-13, Mill Supt., and Asst. Mgr., Bacio G. & S. M. Co., Bacio, Dgo., Mex.

Present position: Mill Supt., Commonwealth Min. & Mill. Co.

Michael T. Williams, Westonia, Western Aust.

Proposed by W. L. Saunders, George A. Howells, L. D. Albin.

Born 1871, Victoria, Aust. Matriculation Standard, Corporate High School, Bendigo, Vict. 1899, Certificate of Competency as Mine Mgr., Bendigo School of Mines, Vict. 1892-1903, Various positions, Johnson's Reef G. M. Co., Bendigo, Vict. 1903-04, Acting Mine Mgr., Johnson's Reef G. M. Co., Vict. 1904-05, Mine Mgr., Watkins Tin Min. Co., N. S. W. 1905-10, Mine Mgr., Arthur's Creek G. M. Co., Vict. 1910-13, Mine Mgr., Central Cookman's G. M. Co., Maldon, Vict.

Present position: 1913 to date; Genl. Mgr., Edna May G. M. Co.

Associate Member

George E. Hayler, Jr., Joplin, Mo.

Proposed by Charles T. Orr, Herbert DeQ. Taylor, C. A. Wright.

Born 1871, Ann Arbor, Mich. 1890-94, Univ. of Mich., B. S. M. E. Associate Member, American Institute of Electrical Engineers. 1894-98, The Gas Electric Light and Power Co., Van Wert, Ohio. 1898-99, Served one year, Spanish American War. 1899-1902, Genl. Mgr., Gas Electric Light and Power Co., Van Wert, Ohio. 1902-04, Engr., Auburn Automobile Co., Auburn, Ind. 1904-05, Engr., The Arnold Co., Chicago. 1905-09, Engr., Supt., Genl. Mgr., The Spring River Power Co., Joplin, Mo.

Present position: 1909 to date; Genl. Mgr., Empire District Electric Co.

Junior Member

Lyle Maxon Barker, Rolla, Mo.

Proposed by C. R. Forbes, G. H. Cox, D. H. Radcliffe.

Born 1894, Hampton, Iowa. 1911, Grad., Fort Pierre High School, Fort Pierre, S. D.

Present position: Student, Missouri School of Mines.

Daniel Boyd, Toronto, Ont., Canada.

Proposed by George A. Guesas, H. E. T. Haultain, Frederick C. Dyer.

Born 1895, Ireland. Toronto Public Schools. 1907-11, High School, Toronto. 1911-12, Toronto Hydro-Electric System. 1913 (summer), Surveying with City of Toronto. 1914 (summer), Inspecting with City of Toronto Testing Lab. 1915 (summer), Assaying, Eustis Min. Co., Eustis, Quebec.

Present position: Student, Univ. of Toronto.

Frederick G. Brenneman, Eureka, Colo.

Proposed by F. W. Traphagen, Harry J. Wolf, William R. Chedsey.

Born 1892, Pottsville, Pa. 1911, Grad., Bellefonte Academy, Bellefonte, Pa.

Present position: Student, Colorado School of Mines.

Robert Heywood Burrage, Needham, Mass.

Proposed by George S. Raymer, H. L. Smyth, Raymond B. Ladoo.

Born 1892, Gardner, Mass. 1909, Grad., Roxbury Latin School, Boston. 1913, Harvard, A. B. 1912, Engr., White Bauxite Co., Forney, Ala. 1913, Asst. Engr., Chino Copper Co., Santa Rita, N. M. 1914, Transitman, Bay State Street Ry. Co. 1915, Operator, Machinist, etc., Nevada Cons. Copper Co., McGill, Nev.

Present position: Student, Harvard Univ.

Wheaton B. Byers, Cambridge, Mass.

Proposed by H. M. Boylston, Albert Sauveur, F. C. Langenberg.

Born 1893, Newton Center, Mass. 1915, Harvard, A. B.

Present position: Student, Harvard College, in Metallurgy.

Rodney S. Cragin, Rapid City, S. D.

Proposed by C. C. O'Harra, T. S. Dunn, Welton J. Crook.

Born 1894, Caldwell, Kans. 1908-11, Caldwell High School. 1914 (summer), U. S. R. S., Newell, S. D. 1915 (summer), Assayer, Rattlesnake Jack Min. & Mill. Co., Galena, S. D.

Present position: Student, S. D. School of Mines.

Seamore A. Crandall, Seattle, Wash.

Proposed by Joseph Daniels, Milnor Roberts, Charles E. Weaver.

Born 1888, Rochester, N. Y. 1905-11 Whitworth Academy, Tacoma, Wash., B. S. 1901, Mucking in mine, Cascade Mts. 1902-05 (summer vacations), in Mining. 1906-07, Prospecting in Alaska. 1907-08, Reynold-Alaska Dev. Co., Latouche, Alaska. 1910-12, Development work on mining property in Alaska. 1914, Cameron-Johnson Co., Valdez, Ala. General experience in management, promotion and development of mining properties.

Present position: Student, Univ. of Washington, Secy-Treas., Pico Free Milling & Mining Co.

Francis R. Curran, Butte, Mont.

Proposed by C. H. Bowman, D. C. Bard, Bancroft Gore.

Born 1892, Walkerville, Mont. 1909, Gonzaga College, Spokane, Wash. 1906-08 and 1910-11, Butte High School.

Present position: Student, Montana State School of Mines.

Fred Alexander Davies, Minneapolis, Minn.

Proposed by W. R. Appleby, Peter Christianson, George J. Young.

Born 1894, Aberdeen, S. D. 1907-11, Central High School, Minneapolis, Minn. 1914 (summer), Timberman, Lincoln Mine, Virginia, Minn. 1915 (summer), Worked for Smoke Commission, Anaconda Copper Mining Co., Anaconda, Mont.

Present position: Student, Minnesota School of Mines.

Paul J. Dennis, Rapid City, S. D.

Proposed by C. C. O'Harra, T. S. Dunn, Welton J. Crook.

Born 1892, Hot Springs, S. D. 1907-11, Hot Springs High School. 1912 (summer), Homestake Min. Co., Lead City, S. D. 1915 (summer), Trojan Min. Co., Trojan, S. D.

Present position: Student, S. D. School of Mines.

Richard Merle Fullaway, Golden, Colo.

Proposed by William B. Phillips, F. W. Traphagen, W. G. Haldane.

Born 1892, Los Angeles, Cal. 1898-1907, Common School. 1907-11, Los Angeles High School.

Present position: Student, Colorado School of Mines.

Murray Garrison, Golden, Colo.

Proposed by William B. Phillips, F. W. Traphagen, W. G. Haldane.

Born 1895, Golden, Colo. 1907-11, Golden High School.

Present position: Student, Colorado School of Mines.

Charles B. Gauthier, Golden, Colo.

Proposed by F. W. Traphagen, Harry J. Wolf, William R. Chedsey.

Born 1892, Detroit, Mich. 1911, Grad., Western High School, Detroit, Mich.

Present position: Student, Colorado School of Mines.

Joseph H. Hill, Rapid City, S. D.

Proposed by C. C. O'Harra, T. S. Dunn, Welton J. Crook.

Born 1892, Easthampton, Mass. 1904-06, Lakewood School, Lakewood, N. J. 1906-08, Rapid City High School, Rapid City, S. D. 1908-12, S. D. School of Mines, Prep. Dept. 1910-14 (summer), Homestake Mine, Lead, S. D. 1915 (summer), August Mine, Landusky, Mont.

Present position: Student, S. D. School of Mines.

William J. Hill, Butte, Mont.

Proposed by C. H. Bowman, D. C. Bard, Bancroft Gore.

Born 1892, Oberlin, Ohio. 1906-10, South Bend High School, So. Bend, Ind.

Present position: Student, Montana State School of Mines.

Kenneth John Hlebnikoff, Reno, Nev.

Proposed by Francis Church Lincoln, E. A. Julian, Tasker L. Oddie.

Born 1892, Khabarovsk, Russia. 1911, South Dakota School of Mines. 1912-13, New Mexico School of Mines. 1914, Travelling experience through the Western Mining Districts.

Present position: Student, Mackay School of Mines.

John Stone Hoffman, Rolla, Mo.

Proposed by C. R. Forbes, G. H. Cox, D. H. Radcliffe.

Born 1892, Leon, Iowa. 1906-10, High School, Van Wert, Iowa. 1910-12, Kemper Military School, Boonville, Mo. 1913 (summer), Milling, Dominion Reduction Co., Cobalt, Ont., Canada.

Present position: Student, Missouri School of Mines.

Eugene Hogan, Butte, Mont.

Proposed by C. H. Bowman, D. C. Bard, Bancroft Gore.

Born 1893, Helena, Mont. 1908-12, Butte High School.

Present position: Student, Montana State School of Mines.

J. Harlen Johnson, Rapid City, S. D.

Proposed by C. C. O'Harra, T. S. Dunn, Welton J. Crook.

Born 1892, Philadelphia, Pa. 1906-10, Central Manual Training High School, Philadelphia, Pa. 1910-12, S. D. School of Mines, Prep. Dept.

Present position: Student, S. D. School of Mines.

William Henry Kamp, Rolla, Mo.

Proposed by G. H. Cox, C. R. Forbes, D. H. Radcliffe.

Born 1892, St. Louis, Mo. 1907-11, Yeatman High School, St. Louis, Mo. 1914 (summer), Buffalo Mines, Ltd.

Present position: Student, Missouri School of Mines & Metallurgy.

Alvin Thomas Krogh, Minneapolis, Minn.

Proposed by W. R. Appleby, Peter Christianson, George J. Young.

Born 1893, Necedah, Wis. 1907-11, South High School, Minneapolis. 1913 (summer), Mucker, Miner, Original Mine and Bell Mine, Anaconda Copper Min. Co., Butte, Mont. 1914 (summer), Mucker, Miller Mine, Aurora, Minn. 1915 (summer), Research Dept., Anaconda Copper Min. Co., Anaconda, Mont.

Present position: Student, Minnesota School of Mines.

Harrison M. Lavender, Golden, Colo.

Proposed by William B. Phillips, F. W. Traphagen, W. G. Haldane.

Born 1889, Scotland.

Present position: Student, Colorado School of Mines.

Leslie A. Lind, Butte, Mont.

Proposed by C. H. Bowman, D. C. Bard, Bancroft Gore.

Born 1892, Butte, Mont. 1907-11, Butte High School.

Present position: Student, Montana State School of Mines.

Thomas T. Lyon, Jr., Butt, Mont.

Proposed by C. H. Bowman, D. C. Bard, Bancroft Gore.

Born 1888, Quincy, Ill. 1908, Culver Military Academy.

Present position: Student, Montana State School of Mines.

Harold Langdon McClelland, Cooksville, Ont., Canada.

Proposed by George A. Guess, H. E. T. Haultain, Frederick C. Dyer.

Born 1890, Cooksville, Ont., Canada. 1903-09, High School, West Toronto.

1909-10, Arts Course, Univ. of Toronto. 1914, Assayer, Cobolt Lake.

Present position: 1913 to date; Student, Mining Course, Univ. of Toronto.

Byron Alfred McCrodan, Belleville, Ont., Canada.

Proposed by George A. Guess, H. E. T. Haultain, Frederick C. Dyer.

Born 1895, Belleville. 1913, Underground work, Cobalt, Ont. 1914, Underground work, Queensboro, Ont. 1915, Deloro Min. and Reduction Co., Deloro, Ont.

Present position: Student, Univ. of Toronto.

Roy H. McHardy, Minneapolis, Minn.

Proposed by W. R. Appleby, Peter Christianson, George J. Young.

Born 1891, Minneapolis, Minn. 1907-1906, Public School, Minneapolis, Minn. 1906-10, High School, Minneapolis, Minn. 1914, Oliver Min. Co. 1915, Anaconda Copper Min. Co.

Present position: Student, Univ. of Minnesota.

Samuel McKirahan, Rapid City, S. D.

Proposed by C. C. O'Harra, T. S. Dunn, Welton J. Crook.

Born 1882, Smith Center, Kans. Common School. 1900-07, Miner. 1907-09, Genl. Mgr., Georgetown M. P. & T. T. Co., Georgetown, Colo. 1909-12, Leasing at Georgetown, Colo. 1912-14, Mgr., Loyal Gold M. & M. Co., Denver, Colo.

Present position: Student, S. D. School of Mines.

Harold Arthur Neustaedter, Rolla, Mo.

Proposed by G. H. Cox, C. R. Forbes, D. H. Radcliffe.

Born 1892, St. Louis, Mo. 1898, Public School, St. Louis, Mo. 1899-1900 Private School, Mexico City, Mex. 1900, Public School, St. Louis, Mo. 1902-03, Private School, Durban, So. Africa. 1903-04, Private School, Boxburgh, So. Africa. 1904-05, Drury Academy, Springfield, Mo. 1905-06, Public School, St. Louis, Mo. 1906-07, Public School, St. Louis, Mo. 1906-07, Public School, Lyon Mountain, New York. 1907-09, Public School, Jersey City, N. J. 1909-12, Jersey City High School. 1915 (summer), Arminius Chemical Co., Mineral, Va.

Present position: Student, Missouri School of Mines.

Richard G. Place, Reno, Nev.

Proposed by Francis Church Lincoln, E. A. Julian, Tasker L. Oddie.

Born 1892, Milwaukee, Wis. 1912-13, Univ. of Southern Cal. 1914, Colorado School of Mines.

Present position: Student, Mackay School of Mines.

Fred S. Porter, Seattle, Wash.

Proposed by Joseph Daniels, Milnor Roberts, Charles E. Weaver.

Born 1890, Mitchell, So. Dak. 1904, Grad., Grade School. 1908, Grad., Hgt. School, Mitchell, So. Dak. 1912, Underground, Bunker-Hill and Sullivan M. & C. Co., Kellogg, Idaho. 1913-14, Underground at mining and 13 months in the engineering office, Alaska-Treadwell Gold Min. Co., Treadwell, Alaska. 1915, Miner and Engr., Butte and Superior Mine, Butte, Mont.

Present position: Student, Univ. of Washington.

Walter Herbert Ralph, Golden, Colo.

Proposed by William B. Phillips, F. W. Traphagen, W. G. Haldane.

Born 1892, Glasgow, Scotland. 1899-1905, MacKay & Sutherland English School, Valparaiso, Chile. 1907-10, Mecheston Castle School, Edinburgh, Scotland. Higher education, Scotland. Intermediate certificate in English, Mathematics, German, Spanish and Sciences. 1910-11, Night School, Southport Technical Institute, Southport, England. 1911-12, Night School, Liverpool Technical Institute, Liverpool, England. First class certificate in heat engines and Applied Sciences. 1910-12, Apprentice to Engineering, Liverpool Gas Works, Liverpool and Bootle, England. 1914, Laborer, Colorado School of Mines. 1915, Timber-helper, mucker, etc., United States Smelt. & Ref. Co., Bingham Canyon, Utah.

Present position: Senior Student, Colorado School of Mines.

John Gay Reilly, Rolla, Mo.

Proposed by C. R. Forbes, G. H. Cox, D. H. Radcliffe.

Born 1895, St. Louis, Mo. 1907, Public Schools, St. Louis, Mo. 1909, Smith Academy, St. Louis, Mo. 1913, Culver Military Academy, Culver, Ind.

Present position: Junior Student, Missouri School of Mines & Metallurgy.

Harold Edmunds Sanford, So. Bethlehem, Pa.

Proposed by Joseph W. Richards, Howard Eckfeldt, W. G. Matteson.

Born 1894, Pomeroy, Wash. 1907-12, Portland Academy, Portland, Ore.

Present position: Student, Lehigh Univ.

Fred W. Sherman, Rapid City, S. D.

Proposed by C. C. O'Harra, T. S. Dunn, Welton J. Crook.

Born 1886, Ascotin, Wash. 1906-11, Grad., International Correspondence School, Civ. Engrg. 1905-06, Gold Reef Min. Co., Giltedge, Mont. 1907-08, North Butte Min. Co., Butte, Mont. 1908, Anaconda Copper Min. Co., Butte, Mont. 1909, Gilmore and Pittsburgh R. R. Co., Armstead, Mont. and Salmon, Ida. 1910, San Pedro, Los Angeles and Salt Lake R. R. Co., Caliente, Nev. 1912, Great Northern Ry. Co., Minot, N. D. 1913-15 (summer), Spotted Horse Mine, Maiden, Mont.

Present position: Student, S. D. State School of Mines.

Elwyn L. Smith, Rapid City, S. D.

Proposed by C. C. O'Harra, T. S. Dunn, Welton J. Crook.

Born 1892, Monument, Colo. 1906-09, Colorado Springs High School. 1911-12, S. D. School of Mines, Prep. Dept. 1911-12 (summer), Homestake Min. Co., Lead, S. D. 1913, (summer) Wasp No. 2 Min. Co., Flatiron, S. D. 1914 (summer), Golden Reward Min. & Mill. Co., Deadwood, S. D.

Present position: Student, S. D. State School of Mines.

Frank Augustus Smith, Golden, Colo.

Proposed by William B. Phillips, F. W. Traphagen, W. G. Haldane.

Born 1894, Los Angeles, Cal.

Present position: Student, Colorado School of Mines.

Chester H. Steele, Butte, Mont.

Proposed by C. H. Bowman, D. C. Bard, Bancroft Gore.

Born 1894, Portland, Ore. 1908-10, Broadway High School, Seattle, Wash. 1910-12, Butte, High School, Butte, Mont.

Present position: Student, Montana State School of Mines.

Francis Lewis Stephenson, Bethlehem, Pa.

Proposed by Joseph W. Richards, Howard Eckfeldt, W. G. Matteson.

Born 1891, Pittsburgh, Pa. 1910, Grad., Haverford School. 1913, Yale, Ph. B.

Present position: Student, Lehigh Univ.

Willis B. Strang, Butte, Mont.

Proposed by C. H. Bowman, D. C. Bard, Bancroft Gore.

Born 1895, Billings, Mont. 1909-1912, Billings High School.

Present position: Student, Montana State School of Mines.

William Harrison Whittier, Seattle, Wash.

Proposed by Joseph Daniels, Milnor Roberts, Charles E. Weaver.

Born 1889, Forks, Wash. 1908 (summer), Highway construction. 1909-11, Highway location and construction, Highway Dept., Wash. 1911-12, Chehalis Co., Cruise, Chehalis Co., Wash. 1912-13, Land and Mine surveying, B. C. 1913-14 (summer), Surveying and placer mining in West Couteauy County, B. C. with Gore & McGregor Co., Victoria, B. C. 1915 (summer), In B. C. for James Lasly & Co. of Chicago, New Orleans, Minneapolis, Portland, Ore. and Seattle, Wash.

Present position: Student, The University of Washington.

S. C. Wong, Butte, Mont.

Proposed by C. H. Bowman, D. C. Bard, Bancroft Gore.

Born 1892, Honolulu, H. I. 1907-11, Territorial Normal School, Honolulu. 1913-15, Michigan School of Mines, Houghton, Mich.

Present position: Student, Montana State School of Mines.

CHANGES OF ADDRESS OF MEMBERS

The following changes of address of members have been received at the Secretary's office during the period Oct. 10 to Nov. 10, 1915. This list, together with the list published in *Bulletin* Nos. 100 to 107, April to November, 1915, and the foregoing list of new members, therefore, supplements the annual list of members corrected to Mar. 1, 1915, and brings it up to the date of Nov. 10, 1915.

ALDRICH, TRUMAN H., JR. 1026 Glen Iris Ave., Birmingham, Ala.
 ANDERSON, ROBERT J. Instr. in Metallurgy, Univ. of Missouri, Rolla, Mo.
 BABB, P. A. Avenida Cinco de Mayo No. 32, Mexico City, D. F.
 BEARD, JAMES T. 126 St. James Place, Brooklyn, N. Y.
 BEATON, NORMAN H., 1005 Washington, Bldg., Third & Spring Streets,
 Los Angeles, Cal.
 BEEKEN, LEWIS L., Carnegie Institute of Technology, Schenley Park, Pittsburgh, Pa.
 BLAYLOCK, D. W., Min. Engr.; Div. Engr., Big Creek Coal Co. of Chicago,
 Harrisburg, Ill.
 BOISE, CHARLES W. . . . Care M. K. Shaler, 4 Bishopsgate, London, E. C., England.
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 via Capetown, Belgian Congo, S. Africa.
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STACK, FRANK LAWRENCE, Care Cerro de Pasco Min. Co.,	Cerro de Pasco, Peru, So. Amer.
STEELE, WALTER L, Vice-Prest. & Genl. Mgr., Mineral Milling Co.,	Hazel Minee, Van Horn, Tex.
STIEBEL, H. J.	84 Main St., Meaderville, Butte, Mont.
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SWART, WALTER G.	807 Sellwood Bldg., Duluth, Minn.
SWEETSER, ARTHUR L, Genl. Mgr Lynnfield Chemical Co.,	Lynnfield, Mass.
THOMAS, CHARLES, S. JR.	Care Mercantile Trust Co., St. Louis, Mo.
THOMAS, GEORGE S.	Care New Park Hotel, Park City, Utah.
THORNHILL, E. BRYANT	Gray Summit, Mo.
TITSWORTH, FREDERICK S.	La Jolla, Cal.
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VRANG, CHRISTON M.	Stanford University, Cal.
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WESTERVELT, E. W.	Ludwig, Nev.
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YOSHIWARA, SHIGETAKE	Dairi Metallurgical Dept., Dairi, Moji, Japan.
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WYMAN, GEORGE H., JR.	Boise, Idaho.
ZELLER, HOWARD PATTERSON	Care Toledo Furnace Co., Toledo, O.

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Name.	Last address of Record, from which Mail has been Returned.
AUER, CHARLES I.	Estacion de Pedricena, Durango, Mex.
BELL, D. A. S.	136 McLaren St., Ottawa, Canada.
BLOW, JOHN J.	172 Rodney St., Brooklyn, N. Y.
BOYS, HARRY R.	c/o Aurora Cons. Min. Co., Aurora, Nev.
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CHAMBERLAIN, JOHN R.	El Aguila Oil Co., Apartado 150, Tampico, Tamps, Mex.
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COLLINS, W. J.	222 Mathilda St., Pittsburgh, Pa.
COOK, PAUL RICHARDSON.	4159 Grand Boulevard, Chicago, Ill.
CRARY, CHARLES N.	Kimberly, Nev.
DOWLER, HARRY P.	Penn Mary Coal Co., Heilwood, Pa.
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EATON, EDWIN R.	Manatawny Bessemer Ore Co., Manatawny, Pa.
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HOLDEN, ROY J.	Blacksburg, Va.
HYDE, JAMES M.	1841 Shattuck Ave., Berkeley, Cal.
JONES, THOMAS J.	Kyshtim Min. Wks., Perm Govt., Russia, via Petrograd.
KERR, D. GILLESPIE.	Confederation Life Bldg., Toronto, Canada.
KIRKLAND, T. C., Cia. Met. y Refinadora del Pacifico, S. A., Fundicion, Son., Me.	
KITSON, HOWARD W., c/o British Columbia Copper Co., Voight's Camp, B. C., Canada.	
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LAWRENCE, T. J.	Topia, Durango, Mexico.
LE NOIR, FRANK H.	Douglas, Alaska.
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MAINWARING, H. M. C.	Chillagoe, Ltd., Chillagoe, Queensland, Aust.
MENEFEE, ARTHUR B.	1440 Fourth Ave., Louisville, Ky.
MILLER, FRANK BARTON.	Blair, Esmeralda Co., Nev.
MORRIS, CHARLES E.	Morris & Flint, Pony, Madison Co., Mont.
NASH, WILLARD H.	297 Delaware Ave., Buffalo, N. Y.
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NOBS, FREDERICK W.	La Leonesa, Matagalpa, Nicaragua, C. A.
NORRIE, WILLIAM G.	Lucky Jim Zinc Mines, Three Forks, B. C., Canada.
PALMER, CORTLANDT E.	2 Rector St., New York, N. Y.
PINKHAM, W. F.	Battle Mountain, Nev.
PORTER, ROBERT S.	c/o Fortifications, Culbra, Canal Zone.
PRETTYMAN, FRANK REMINGTON.	Oyster Bay, L. I.
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REYNOLDS, L.	Box 857, Nelson, B. C., Canada.
RHODES, WILLIAM B.	Golden, Colo.
ROBINSON, HARRY O.	Calle Madrid 15, Mexico City, Mex.
RODRIGUEZ, JUAN C.	Apartado 87, Saltillo, Coahuila, Mex.
SALE, ANDREW J.	Giroux Cons. Mines Co., Kimberly, Nev.
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SEIBERT, PERCY A.	Hagerstown, Md.

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WRIGHT, JESSE T.	Wallace, Idaho.
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YOUNG, C. D.	Penn. R. R. Co., Altoona, Pa.

NECROLOGY

The deaths of the following members were reported to the Secretary's office during the period Oct. 10 to Nov. 10, 1915:

Date of Election.	Name.	Date of Decease.
1900	**Baggaley, Ralph	Sept. 23, 1915.
1910	*Blackmar, Frank H.	Sept. 29, 1915.
1900	*Nack, Charles	Sept. 27, 1915.
1914	*Cobeldick, W. Morley	Oct. 1915.
1887	*Hammond, Isaac B.	Sept. 21, 1915.
1906	*Hoffmann, John I.	Sept. 15, 1915.
1915	*Nelson, Carl N.	Aug. 14, 1915.
1912	*Parsons, Roscoe R. S.	Aug. 27, 1915.

Isaac B. Hammond was born May 24, 1844, at Lamb's Corners, N. Y. During his childhood, the family removed to Kankakee, Ill., and at the age of 23 Mr. Hammond pushed still farther west, and entered the service of the Union Pacific R. R., in the construction department of which he served until the transcontinental line was completed. He then went to Deadwood, and was engaged for many years in the manufacture of stamping and other machinery used in the production of gold. In 1888 he went to Alaska, whence he was sent to London to purchase a 120-stamp mill for use in the notorious Bear's Nest mine, near Douglas. After the collapse of that enterprise, Mr. Hammond bought the machinery and brought it to Portland, Ore., where he organized the Union Electric Power Co. and, a year later, the Hammond Manufacturing Co., of which he was President for 25 years. In 1911 he retired from active business, and on Sept. 21, 1915, he died, after three weeks' illness, from arterial sclerosis, complicated with other diseases.

As one of the pioneer manufacturers of Portland, Mr. Hammond had much to do with the progress of that city, and was at one time President of its Board of Trade. He became a member of the Institute in 1887.

Roscoe R. S. Parsons, a member of the Executive Committee of the St. Louis Section of the Institute, and a member of the Institute since 1912, died on Aug. 27, 1915, at the close of a month's vacation on the Pacific Coast. Apparently in good health, Mr. Parsons was sud-

denly attacked with acute indigestion that in a few hours terminated his life.

Mr. Parsons was born at Bonne Terre, Mo., on Oct. 5, 1875, and spent his years with the St. Joseph Lead Co. His father, Charles B. Parsons, was the first superintendent of this now famous company, and on his shoulders fell the task of building up and successfully conducting a business confronted by various financial and technical difficulties requiring unusual courage and determination.

Mr. Parsons returned to Bonne Terre and entered the employ of the company as his father's assistant after taking the literary course at Princeton University. Upon the death of the latter a few years ago, he became the general superintendent, more recently and since the consolidation of that company with the Doe Run Lead Co. having served as vice-president and resident director of the St. Joseph Lead Co.

Mr. Parsons had a keen interest in the local affairs of Bonne Terre, a mining town of about 5,000, was an active worker in building up and improving social conditions there, and was very popular with the employees of the company. Although always vested with authority, his great influence with the miners gained for him a fellow feeling and he was regarded rather as the Big Brother of the community, endearing himself to all by his kind disposition, gentle, thoughtful and lovable character. Fortunate and blessed indeed are they who depart from this earth leaving behind such tender memories, and the manifest love and respect that he won from those who knew him.

He is survived by a wife and daughter.

H. A. WHEELER.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at a meeting of the Institute. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author.

Protecting California Oil Fields from Damage by Infiltrating Water

BY R. R. MCLAUGHLIN,* SAN FRANCISCO, CAL.

(San Francisco Meeting, September, 1915)

IN most branches of the mining industry it is a well-recognized fact that care must be taken to protect the mineral deposit from undue physical injury. It is comparatively easy to grasp this idea when the mineral is a solid and the lives of workmen depend directly upon the care with which it is removed from its natural position, but when the product is a liquid drawn up from pools several thousand feet beneath the surface of the ground through openings only a few inches in diameter, it requires some imagination to picture underground conditions and the changes which must often occur. The development of the petroleum deposits of California has been attended by numerous accidents, the results of which are readily noticed at the top of the wells and in the profit and loss accounts of the operators. Determination of the cause of these accidents has been the subject of considerable study, which naturally has fallen upon men familiar with technical methods.

The greatest damage to the oil deposits of California has resulted from water finding its way into the oil-bearing sands, and it is the purpose of this paper to set forth briefly this phase of the oil industry. Much of the material here presented has previously appeared in publications issued by the California State Mining Bureau, which, under the direction of State Mineralogist Fletcher McN. Hamilton, has investigated the subject and taken steps toward improvement of conditions. The writer, having been in charge of this branch of the Bureau work, has gained considerable knowledge of existing conditions.

Generally speaking, the oil-bearing formations of California are soft, unconsolidated sand beds, separated by clay strata, which are also soft. Some of the sand beds carry water instead of oil. All of the beds are more or less separated, probably being of lenticular shape. Very few of the beds in the oil fields lie flat or undisturbed over any great area. Folding and tilting are common, and faulting also occurs in some of the fields. The accumulation of oil is not confined to any single structural feature, such as the anticline, but occurs in many other structural

* State Oil and Gas Supervisor.

features. Water-bearing strata occur in practically every oil field in the State. Complete data are not at hand for a comparison of the damage done in the various fields. These water sands are sometimes above, below, and between the oil sands. When water and oil sands are both penetrated by a well, the water enters the oil sand. The water frequently forces the oil back into the sand so that pumping produces nothing but water. The water not only damages the well which it first enters, but affects neighboring wells, causing their product to become partly or entirely water. The probability of such damage is usually foreseen. The method of preventing damage from this source is to insert screw pipe (or casing) into the well. It serves as the lining or wall of the well, and if carried through and below the water sand and firmly landed in some sort of impervious stratum, such as clay, it prevents the water from leaking down into the well. Sometimes it is necessary to pump cement in behind the casing in order to seal thoroughly the passages down which the water might otherwise run. Several such lines (or strings) of casing are sometimes placed inside the first one until together they resemble a telescope with the small end at the bottom of the well, in the oil sand.

The protection of a well from water intrusion comprises two operations: First, determining where the casing should be landed; and, second, the actual work of placing the casing in the well in a workmanlike manner. These operations are interdependent, and either is useless without the other.

Some idea of the financial loss of the oil operator, when water is not controlled, may be had from the following instances:

In the Coalinga field on a certain property is a well about 2,700 ft. deep. It was drilled three or four years ago at a cost of about \$30,000. Initial production was over 200 bbl. of oil per day and the total production about 100,000 bbl. of oil, giving a profit of possibly \$10,000 over regular operating expenses, but not including the cost of the well. About two years ago water stopped this production entirely. The owners are financially unable to repair the well at present, and each day adds to the damage being done to surrounding property.

In a portion of the Midway-Sunset field there are two known oil sands, both productive. There are water-bearing sands above the upper oil sand and also between the two oil sands. When a well is drilled into the lower sand two extra strings of casing (about 2,000 ft. of 8-in. and 2,500 ft. of 6-in.) should be inserted to keep the water out of the upper sand. The expense for extra casing and labor may amount to \$10,000 per well. Some operators have neglected or refused to protect their wells in this manner, which has resulted in serious damage to the wells of other operators which depend upon the upper sand, because they are of too small diameter to allow deeper drilling.

An approximate idea of the ultimate loss that will ensue, unless the top sand is protected, may be gained from the following figures: A single well, at present producing, has during three and a half years produced almost 300,000 bbl. of oil. Other wells, though not so large, should produce for a much longer time than three and a half years, and 300,000 bbl. each is not an unreasonable estimate of their total production. The upper sand extends over about 400 acres, and at 5 acres per well should supply 80 wells, or a total of not less than 24,000,000 bbl., or \$12,000,000.

A minor instance which came under notice is typical of much inefficient work that has been done. A superintendent, who had previously finished several wells satisfactorily with advice from a geologist, proceeded without such advice and drilled another well too far through the oil sand and encountered a strong flow of water. This "bottom water" was encountered within 2 ft. of where it had been expected by the geologist, from his study of logs of surrounding wells. Advice would have been to stop drilling about 15 ft. above the point to which it was carried. A contractor was engaged to cement the bottom of the hole after the flow of water had been checked by a "bridge." Even if the first cementing operation proved successful, the needless expense on this well probably amounted to about \$500. The chief aim of the geologist or engineer is to meet such circumstances *in time to prevent trouble*, rather than to aid in patching up improperly drilled wells, which procedure is, at best, apt to be troublesome and unsatisfactory.

It must be borne in mind, of course, that the natural oil reservoirs are exhaustible and as the oil is removed water is apt to take its place. Some operators are prone to assume that the presence of water in an oil well is perfectly natural and that infiltration could not have been avoided. It has been proved that this hypothesis is not always correct, for in many cases where water has appeared, the correction of faulty material or workmanship has caused the water to disappear.

There are at least three causes for water entering oil wells: (1) failure of casing, (2) faulty drilling operation, and, (3) natural replacement of oil by water as the oil is pumped out. There are also combinations and variations of these causes, modified and accelerated by movements of the ground itself.

The accompanying sketch (Fig. 1) of the bottom of four wells in the Coalinga field illustrates some of the underground conditions and the methods of recording and presenting the data. The wells were drilled in order as numbered. The strata are not bent or deformed in this particular locality. In each well the 10-in. casing was intended to shut off the water occurring above the oil sands, and in well No. 3 it was landed at a depth of 1,880 ft., in conformity with successful results at No. 1 and No. 2. However, the sand between 1,980 ft. and 2,016 ft. was found to

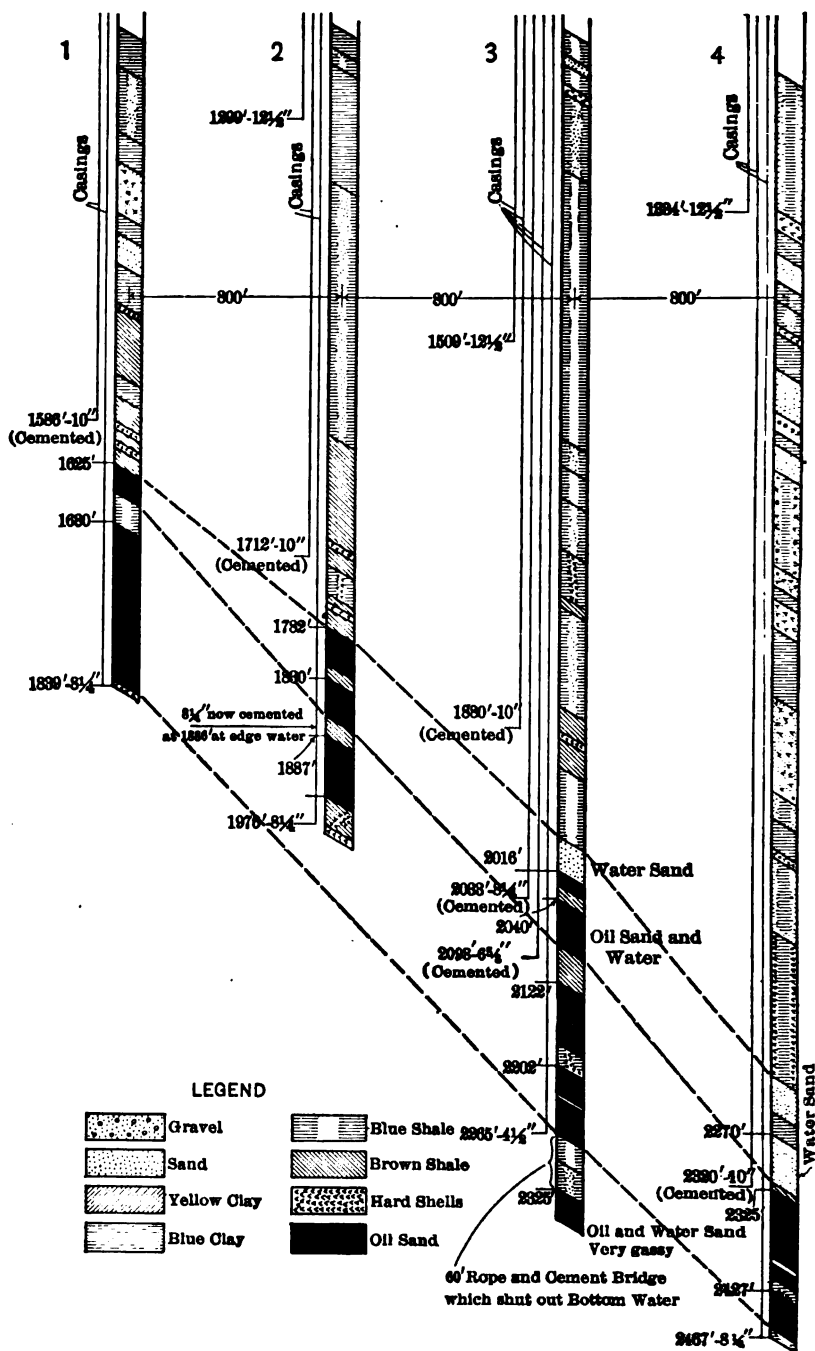


FIG. 1.—CROSS-SECTION OF LOWER PART OF FOUR WELLS IN THE COALINGA OIL FIELD, CALIFORNIA.

contain some oil and considerable water, hence it was necessary to use more casing to shut off this water. Since some experimenting was necessary, two strings, the $8\frac{1}{4}$ in. at 2,033 ft. and the $6\frac{5}{8}$ in. at 2,098 ft., were used. Had all conditions been known beforehand only one string would have been necessary, but when the $8\frac{1}{4}$ -in. casing was landed and the well drilled deeper more water was encountered and the $6\frac{5}{8}$ -in. casing had to be used. The final oil string of casing, $4\frac{1}{2}$ in. in diameter, was carried to 2,351 ft. and upon testing, the well was found to produce mostly water, so the bottom was plugged (using waste, rope, iron lathe cuttings and cement) up to 2,265 ft. and the well again tested, when it was found to be free from water.

Since well No. 3 was one of the first wells in this locality to encounter water in the oil sands, where oil was expected, careful work was necessary on the next well drilled, to check the operations. This was done on neighboring wells, not shown on the drawing, and since similar results were found, when well No. 4 was drilled the top sands were shut off with the 10-in. casing at 2,320 ft. The well was drilled to 2,467 ft. only, so as not to encounter the sand which contained water in well No. 3 between the 2,325 and 2,351 ft. levels.

After completion, all these wells produced clean oil for about a year, and then the No. 2 began to show a little water, and about three months later showed about 80 per cent. There were three possible sources of the water: From other wells through the oil sand, or from above the oil sand, or from the bottom sands penetrated. The drawing shows that the possibility of bottom water was slight. The possibility of water from above was tested by removing the $8\frac{1}{4}$ -in. "oil string," putting a "bridge" or plug in the open hole between the bottom of the 10-in. "water string" and the top of the oil sand. After bailing the well dry above the plug no water came in, proving that the "water string" was not leaking at any point and was effectively excluding upper water. Therefore the water in No. 2 was probably coming through the oil sand itself, and as the same sand showed both oil and water between 1,980 ft. and 2,095 ft. in No. 3 it was concluded that the water was rising along the dip of that sand as the oil was removed. Such a movement is called the encroachment of "edge water." The $8\frac{1}{4}$ -in. casing was again put in and cemented at 1,886 ft. in the shale presumed to correspond to that found at 2,091 to 2,122 ft. in well No. 3, and 2,318 to 2,325 ft. in well No. 4. After pumping the well for about 10 days, the oil was found to contain less than 1 per cent. water, proving that water had been coming through the upper sands, between 1,782 and 1,870 ft. After about four years the water has commenced to show in well No. 1. This movement naturally follows the removal of oil and cannot be stopped, but the particular sand affected can be kept separated from other productive sands.

As in all other important industries involving the winning of some

mineral product from the earth, a thorough plan should be mapped out, and followed or revised in the light of new developments. In other words, there should be a thorough geologic investigation. Some large operators realize this and have facilities for thorough geologic work at all times. Other large concerns give but little attention to the subject. Probably the oil industry is in this respect not so well managed as the metal-mining industry. Small operators frequently feel that they cannot afford the expense of geologic work.

Several hundred oil companies are operating in California and in the past no concerted effort has been made by them to develop their lands systematically or to control the water trouble. Constant agitation of the subject has resulted in the passage of a law (Chapter 718, Statutes of 1915) framed by the State Mining Bureau, at the invitation and with the assistance of many oil operators. The law became effective Aug. 8, 1915. As it is along original lines, a brief outline may be interesting, together with some remarks upon the aims of its sponsors.

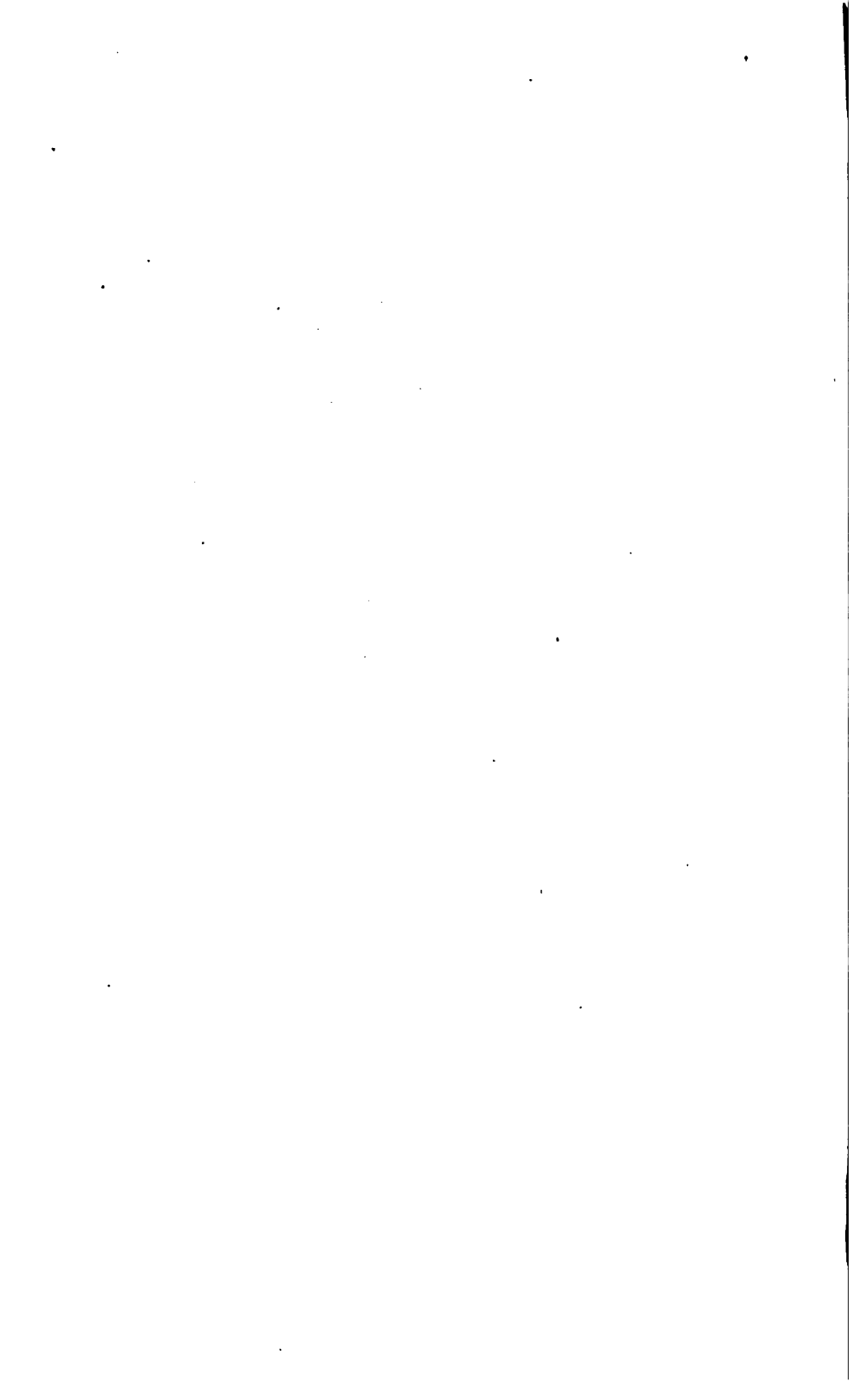
The law gives the State Mining Bureau supervision over this work, the expense being met by assessments upon the producers. The work is directed by an officer designated as State Oil and Gas Supervisor, who appoints four deputies; all of these officials must be either engineers or geologists. The Supervisor's headquarters will be at the State Mining Bureau in San Francisco and the deputies will reside in the various oil fields. Operators are required to file all records of work done on wells and other pertinent information. The records are to be considered as confidential unless released by the operator furnishing them. The Supervisor is empowered to order repair work done on wells and if it is not done he can place agents on the ground to do the work. The cost of the repair work becomes a lien upon the property. Abuse of power by the Supervisor is prevented by making his orders subject to review and reversal by an arbitration board appointed by operators.

The law is rather lengthy and explicit, but no rigid construction can be expected to solve such varied problems as exist in our various oil fields. It is believed that operators generally will coöperate with the officials in the endeavor to make the law fit economic and physical conditions.

The underlying purpose of the framers of the law was to establish a technical department that would gather all the facts and present conclusions to those having charge of drilling operations. In other words, it is hoped that the department will fill the place of a consulting engineer. The need of such a position has been recognized by some operators in their own private organization, while others have not realized its value or have been unable to bear its expense. It is therefore evident that the first work of the State Mining Bureau must be educative. Every man engaged in the work must primarily be able to touch the chord of human interest and present the technical facts to his audience, the operators, in

such a fashion that they can see the subject as clearly as he does. Technical workers have frequently complained that their work is not given proper recognition by the public, and in many cases this has been due to the fact that no effort was made to grasp the thread of public thought on that special subject and turn it into the proper channels. The legislation here considered presents one of the best opportunities for those administering it to demonstrate whether or not technical men are able to take hold of a large problem and present it in intelligible form to men not technically educated but thoroughly skilled in drilling deep and difficult wells.

Not alone to the oil operators, but also to the public which is interested in conservation of the oil, as well as all other natural resources, is the Bureau under obligation to solve the problem outlined in this paper. California drillers have already proved their ability to cope successfully with the most severe physical conditions and will undoubtedly be found willing to assist the engineer who now has an opportunity to demonstrate his usefulness.



DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Notes on Flotation*

BY J. M. CALLOW,† SALT LAKE CITY, UTAH

(New York Meeting, February, 1916)

HISTORICAL SKETCH

THE selective action of oils for lustrous minerals was first disclosed by Haynes in 1860. In 1885, Miss Carrie Everson elaborated this idea and also disclosed the fact that acid increased the so-called selective action. Her patent called for oils, either animal, vegetable or mineral, also for an acid or salt. The process was tried out on a practical scale in Baker City and Leadville, in 1889, and failed: first, because, as has since been shown, of the unsuitability of the ore to flotation; and second, because the invention was too far in advance of the times. Then followed the Elmore brothers, first with their bulk oil processes and later with their vacuum method. The basic principles of oil flotation were undoubtedly covered by the above inventors, the work done since their time being merely a building up on ground work laid by them. Different kinds of oil, different quantities of oil, and all the varying degrees of agitation, were all exemplified and practiced by them in one phase or another, and the developments that have since been made are but elaborations of the fundamental principles laid down by Haynes, Everson, and the Elmore brothers.

In 1902, the Potter-Delprat process was developed in Australia. In this process no oil was used, the mineral being raised by the generation of gas, brought about by the introduction of acid into the pulp. The mineral particles appeared on the surface of the separatory vessel, in the form of a scum or froth, buoyed up by minute gas bubbles attached to them. This no doubt first gave the suggestion of gaseous flotation.

In 1902, also, Fromont, an Italian, was granted a patent in which he combined violent agitation with oil and gaseous flotation, the gas being generated within the pulp, in much the same way as in the Potter-Delprat process.

In the same year, Cattermole came out with a unique method. First, the pulp was emulsified with a small quantity of oil, by violent

*Originally presented at the annual meeting of the Utah Section, Salt Lake City, Oct. 4, 1915.

† Consulting Engineer.

agitation, and then submitted to a slow stirring action in a second machine. This granulated, or coagulated, the minerals that had been oiled into nodules which were afterward separated from the pulp by gravity. The defect of this process was that only part of the mineral granulated, the rest of it appearing on the surface of the pulp as a scum or froth, and so was lost in the tailings. This defect of the Cattermole process suggested the fundamental idea of the process afterward described by Sulman. Picard, and Ballot in their patents, in which, instead of granulating part of the mineral, all of it was floated. This patent forms the basis of all the Minerals Separation operations. It was first exploited in Australia and in a short time replaced all other flotation processes in that country.

In 1904, MacQuisten brought out his ingenious tube process, which gave excellent results on the sandy portions of the feed, but was inoperative when slime was present. This was a strictly surface-tension method, and its inability to handle slime was a serious limitation.

In 1912, Hyde introduced a modification of the Minerals Separation process into the mill of the Butte & Superior Co., Butte, Mont. This differed from the regular practice in that roughing and cleaning of the concentrates were done.

DESCRIPTION OF CALLOW PNEUMATIC FLOTATION PROCESS

Early in 1909, I did a great deal of work with the MacQuisten flotation process, and was instrumental in the tube-plant installation at the Morning mill at Mullan, Idaho. This work was followed by a large amount of experimenting on the different kinds of existing flotation processes, the outcome of which was the development of the pneumatic process.

The first application of pneumatic flotation for the treatment of ore was made at the mill of the National Copper Co., at Mullan, Idaho, which was designed and built by me. Construction was started on Aug. 14, 1913, and the plant went into operation about Apr. 10, 1914. The flow sheet is shown in Fig. 1. It was a metallurgical success in every way from the start.

Since that date, the process has been adopted by nearly all the other mills in the Coeur d'Alenes treating lead and lead-zinc ores, notably the Gold Hunter, Morning, Hercules, Bunker Hill & Sullivan, Caledonia, Last Chance, Hecla, Standard, etc., a total of about 50 cells in all, treating from 1,500 to 2,000 tons of slime and fine sand feed per day. The same process too has since been adopted by the Inspiration Copper Co., the Arizona Copper Co., the Anaconda Copper Mining Co., the Magma and other copper companies, and by the Silver King, Daly-Judge, Duquesne, and El Rayo mining companies, on lead, zinc and other ores, making a total of some 680 cells operating, or in the course of installation, having a combined capacity of about 25,000 to 28,000 tons per day.

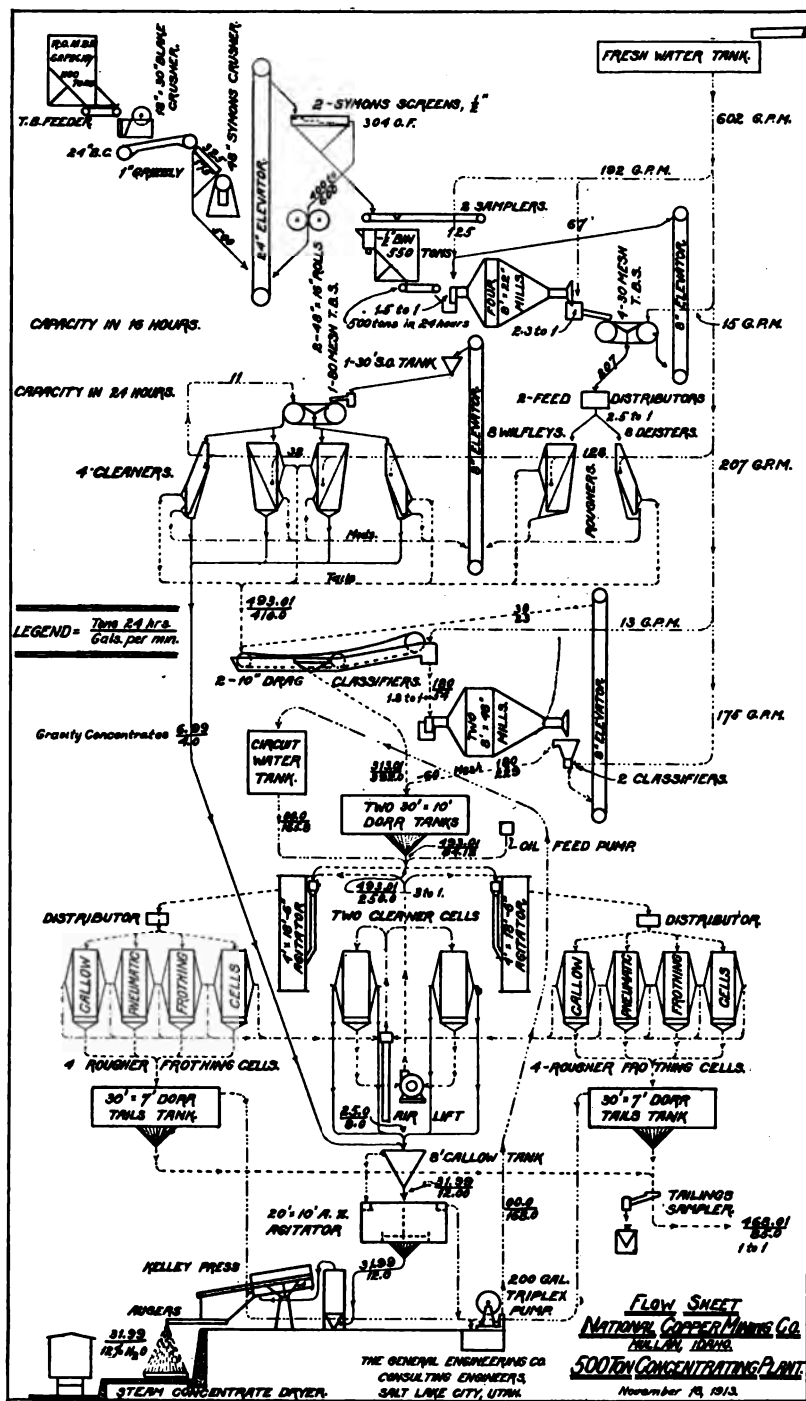


FIG. 1.

Fig. 2 illustrates the various elements composing the Callow process in general.

Mixing

In the mixer *A*, operated by compressed air, the oil, air, and water are mixed and emulsified, the same type of apparatus being in common use in cyanide works. In cases where the oil, or frothing agent, can be fed into the crushing machine or tube mill, this mixer, or Pachuca tank, can be dispensed with, the tube mill discharging direct to the separatory cell.

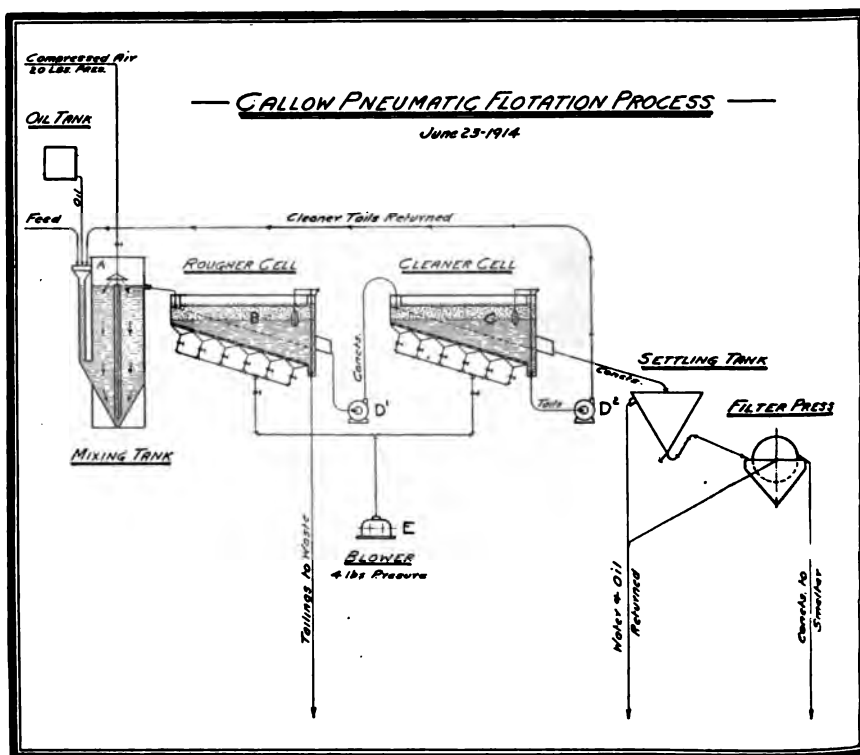


FIG. 2.

It has been conclusively proved that agitation *per se* is not necessary to successful flotation by the pneumatic method. In one of the plants a Pachuca mixer for each four roughing cells was installed. The pulp from a Dorr thickener was delivered to the Pachuca by a belt-and-bucket elevator, the oil being fed into the boot of the elevator. It was found that the mixing obtained in the elevator alone gave as good results as when the Pachucas were used; so the use of Pachuca tanks in this plant was abandoned.

The Separatory Cell

The initial or roughing separatory cell *B* consists of a tank about 9 ft. long overall, and 24 in. wide, with a bottom inclined at from 3 to 4 in. to the foot; it is 20 in. deep at the shallow end and 45 in. deep at the other end. It may be built of either steel or wood, but wood construction is preferable.

Fig. 3 shows the cell in detail. The bottom of the tank consists of a porous medium made of four thicknesses of loosely woven canvas twill, properly supported by a backing of perforated metal to prevent its bulging when under air pressure. Through this porous medium compressed air is forced by the blower *E* (Fig. 2). Porous brick, or any other ceramic material that will give the necessary fine subdivision to the air, may also be used. Some of these have been tried out, but for practical and mechanical reasons the loosely woven canvas twill seems to serve all purposes better than anything else, and has been adopted as the standard porous-bottom construction.

The space underneath this porous medium or bottom is subdivided into eight compartments, each connected by an individual pipe and valve with the main air pipe. By this means the air pressure to each compartment can be regulated (by throttling the valve) to correspond to the varying hydraulic head within the tank, and so as to discharge a uniform amount of air throughout the length of the bottom and maintain a uniform aeration of the contents. A pressure of from 4 to 5 lb. is generally used, each square foot of porous medium requiring from 8 to 10 cu. ft. of free air per minute.

Each longitudinal edge of the tank is provided with a lip and an overflow gutter for the reception of the froth to be discharged. The lower end of the tank is furnished with a spigot discharge fitted with a plug valve, operated by a float, to maintain a uniform water level within the tank and thus, in turn, maintain a uniform and constant discharge of froth under all the varying conditions of feed supply incident to practical milling operations. The water level may, of course, be varied, but is usually maintained at about 10 to 12 in. below the level of the overflow lips.

The tailing is discharged through the spigot and the frothy concentrate is conveyed by means of the side gutters to the pump *D*¹ and thence to the cleaner separatory cell *C*. This cleaner cell is a machine of the same construction as the rougher. In operation, however, it is usually run with a lower air pressure than that used on the rougher. The tailing from the cleaner is returned by pump *D*² to the original feed, a closed circuit thus being maintained on this portion of the feed. The concentrate from the cleaner is the shipping or finished concentrate. Pump *D*¹ can well be eliminated by setting the cleaner at a lower elevation and

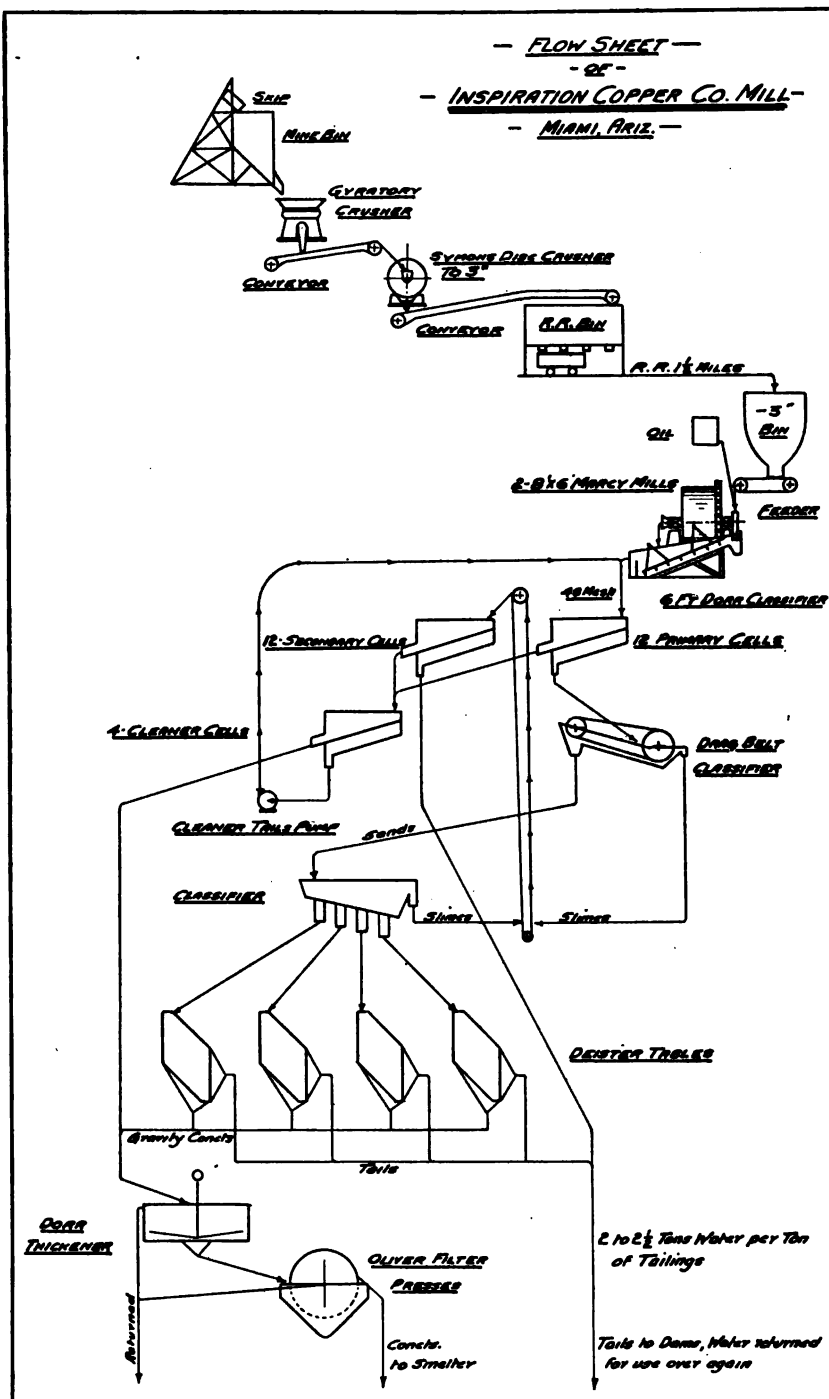


FIG. 4.

conveying the rougher froth to it by gravity. Usually one cleaner serves four roughers.

Cells May Be Used in Parallel or Series

The machines may be run either in parallel or in series without any sacrifice in the capacity for a given number of cells. Recent experience goes to show that on some ores the series treatment gives a slightly cleaner tailing; on others it does not. It is not necessary to extend this arrangement of cells beyond two cells in series. In a heavily mineralized ore this arrangement is decidedly advantageous, and in such a case the rougher concentrate might be of high enough grade to dispense with the re-cleaning operation. The froth from the second cell in the series might be returned to the original feed in the same way that the tailing is returned from the cleaner when practicing a roughing and cleaning operation. A number of such combinations are possible.

At the mill of the Inspiration Copper Co. (Fig. 4), the original feed goes to 12 primary roughers, the tailing from which is classified into sand and slime, the sand going to tables and the slime to 12 secondary roughers. The concentrates from both the primary and secondary roughers go to four cleaner cells, and the tailing from the cleaner cells is pumped back into the circuit.

Froth Formation

The froth is generated as the result of injecting the finely divided air into the bottom of the already emulsified pulp; it continues to form and to overflow so long as it is furnished with pulp of the proper consistency, adequately mixed with the right quantity and kind of oil or frothing agent. Measured from the water level within the tank, the froth produced may be from 14 to 16 in. thick and will be more or less voluminous, coarse or fine grained, dry or watery, according to the character of the ore and the kind and quantity of oil introduced. The condition of the froth may be varied therefore, by changes in the kind and quantity of oil used, and the quantity of air injected.

In some ores, rich in sulphides, and where a comparatively low-grade concentrate will suffice, the cleaning cells may not be necessary, but on low-grade ores having a high ratio of concentration and when a concentrate of extreme cleanness and of maximum grade is required, a cleaner is desirable.

Pulp Density

The pulp to be treated may be of varying density, from $2\frac{1}{2}$ of water to 1 of ore, up to 5 or 6 to 1. For a mixture of sand and slime, the former ratio is preferable, but for a pure slime mixture (minus 200 mesh) the

larger proportion of water is allowable. The particular density is not a matter of as much importance as the supplying of pulp of uniform density, since each variation in the density of the pulp requires a readjustment of the oil supply, the quantity of oil increasing in proportion to the increased volume of the pulp, independent of its solid contents.

Capacities

A normal capacity for each standard roughing cell is 50 tons per 24 hr. This, of course, will vary according to the nature of the ore. In one plant which practices gravitation previous to flotation, only the fine sand and slime being treated, the rate is 50 tons per rougher cell. The Inspiration Copper Co. practices flotation as the prime process, each 800-ton section employing 24 roughing cells and four cleaners. The cells in this case are run in series, the primary cells treating the original feed and the secondary cells treating only the slime from the primary tailing after the sand has been removed. This gives an average of 33.3 tons per roughing cell. The Arizona Copper Co.'s plant (Fig. 5) will treat the slime and re-crushed sand from previous gravity treatment; out of an original tonnage of 4,000 about 3,600 tons will be treated by flotation. This will be handled on 63 roughers run in parallel, and 18 cleaners, or an average of approximately 57 tons per roughing cell, or 45 tons per cell for roughing and cleaning.

Some tests have shown little difference in recovery, whether running 45 tons to the cell or 65; but that the recoveries commence to decline as soon as the tonnage exceeds 75 tons. In the Coeur d'Alenes, on zinc-lead ores, 35 tons per cell is an average capacity. The flow sheet of the Daly-Judge mill (Fig. 6) shows the arrangement for treating a zinc-lead ore in the form of a fine sand and slime feed.

Oils Used

The oils used may be broadly divided into "frothers" and "collectors." The pine oils are good frothers and coal tar and its various subdivisions are good collectors. On some ores, crude pine tar will in itself combine the properties both of frothing and collecting; on others, this may have to be enriched by the addition of some one of its more volatile constituents, such as refined pine oil, turpentine, or wood creosote.

Generally speaking, the coal-tar products are poor frothers, and to get a sufficient volume of froth to insure a high recovery it is often necessary to add refined or crude pine oil, creosote, etc. At the Inspiration mill, for instance, the mixture is 80 per cent. crude coal tar, 20 per cent. coal-tar creosote; at another plant on similar ore 45 per cent. El Paso coal tar, 40 per cent. coal-tar creosote, 10 per cent. creosol, and 5 per cent.

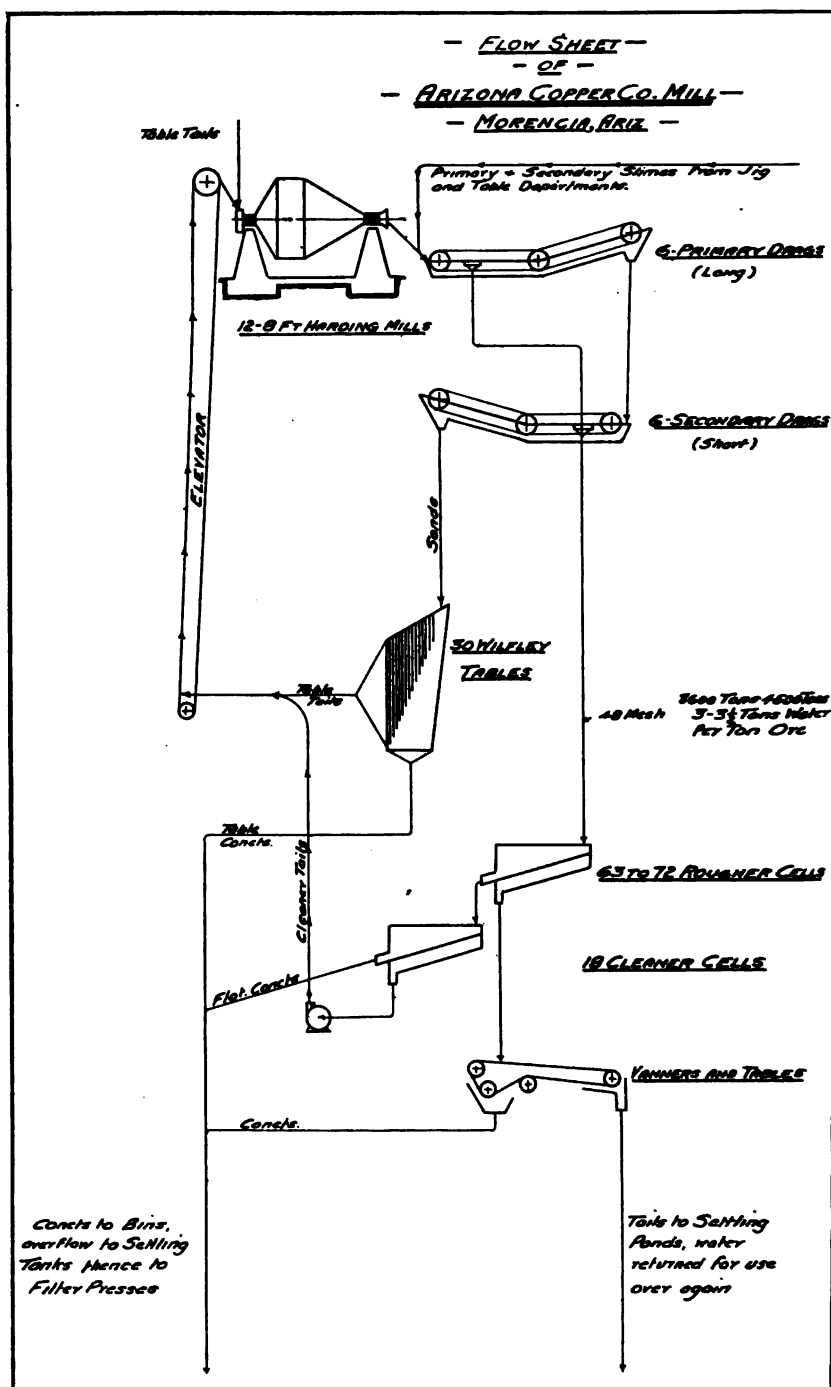


FIG. 5.

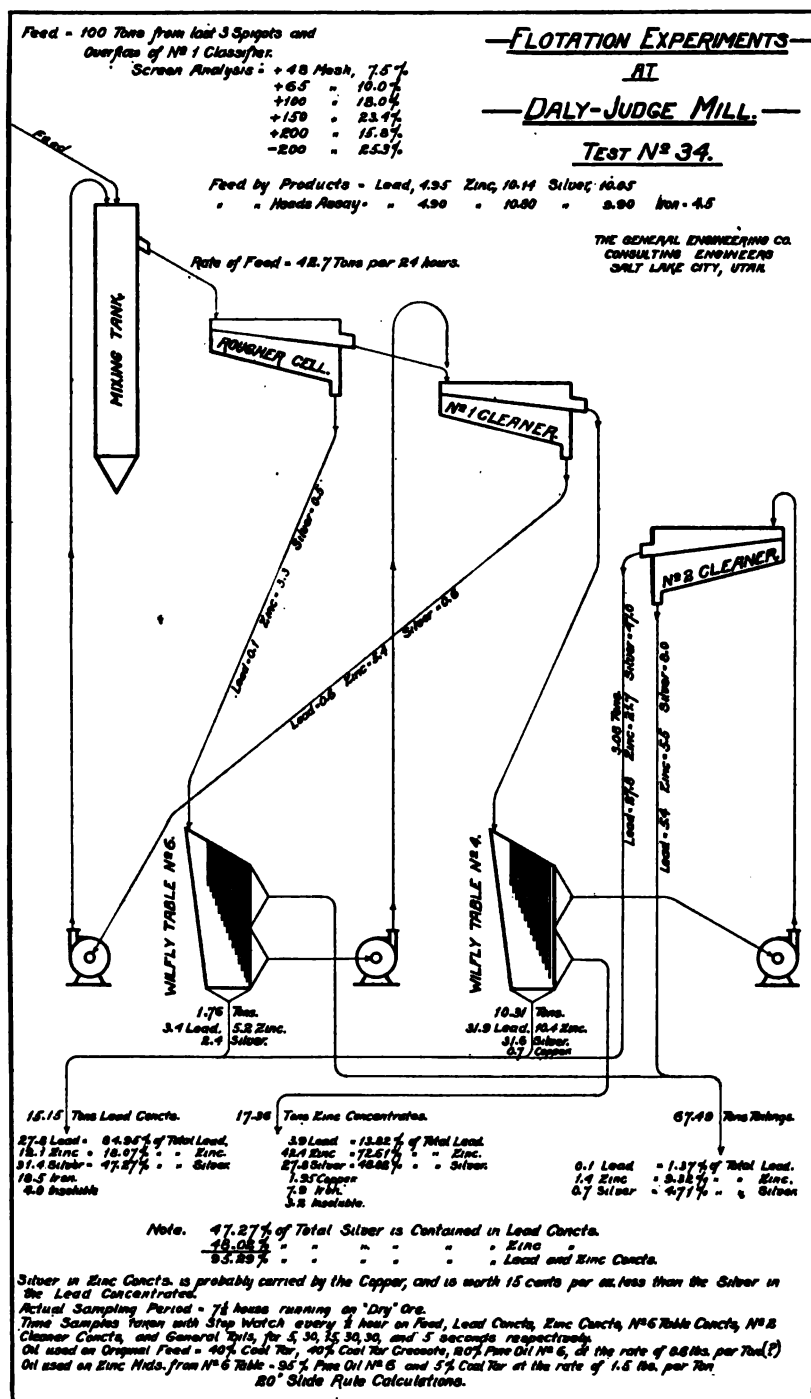


FIG. 6.

pine oil. At Daly-Judge a mixture of 40 per cent. crude coal tar, 40 per cent. creosote, and 20 per cent. pine oil was used. In the Coeur d'Alenes on zinc ore straight wood creosote was used; on the ore of the National Copper Mining Co., plain turpentine will work, but pine oil is better.

At Inspiration, from $1\frac{1}{2}$ to 2 lb. of the mixture per ton of ore was used; at Daly-Judge, 1 to $1\frac{1}{2}$ lb.; and at the National 0.3 lb. of oil is sufficient. In the experimental work at another plant the oil consumption was approximately 1 lb. of mixture per ton; but since the entire plant has been in operation, and the circuit water reclaimed and used over again, the oil consumption has dropped to 0.35 lb. At present the proper kind or kinds of oil and the requisite quantity can only be determined by experiment; no scientific way has thus far been found.

Character of Froth

The froth made by the pneumatic process has the distinctive characteristic of being unstable or ephemeral; that is, it quickly dies when removed from the action of the injected air. The bubbles composing the froth are generated under a hydraulic pressure varying from 15 to 40 in.; upon rising above the water to the froth level, the bubbles burst by reason of the lower pressure of the atmosphere. On bursting, they release the mineral attached to them, which is caught up by the bubbles following immediately behind. The stability of the bubbles depends to some extent, upon the oil used and the nature of the gangue in the pulp treated. Pine oil makes a brittle froth which dies immediately on arriving at the surface. Creosote and light oil make a more elastic envelope which at times will expand into bubbles 3 to 4 in. in diameter before bursting. Pine-oil bubbles are rarely over $\frac{1}{4}$ or $\frac{1}{2}$ in. in diameter. Castor oil, olive oil, candle-makers' oil (oleic acid), palm oil, sperm oil, and other oils of a lubricating nature, have in general been replaced by oils more or less soluble or miscible in water—such as turpentine, pine oil, and all the coal- and wood-tar distillations. The extremely volatile oils such as naphtha, gasoline, ether and alcohol, seem to be of little use except as means for making the pitchy ingredients of coal and wood tars more soluble or miscible.

A large, coarse, and elastic bubble seems necessary to the recovery of coarse-grained mineral, but for the very fine and colloidal mineral, a small and comparatively brittle bubble is necessary.

Power

The National Copper Mining Co., using approximately 950 cu. ft. of air at 4-lb. pressure, and treating 500 tons per day on eight roughers and two cleaners, required 35 hp.; this shows a requirement of 3.5 hp. per cell, equivalent to 12.53 tons per horsepower or 1.25 kw.-hr. per ton.

Another company using approximately 9,600 cu. ft. of air at 5-lb. pressure and treating 2,400 tons per day on 48 roughers and 12 cleaners, requires 210 hp., equivalent to 3.5 hp. per cell, 11.45 tons per horsepower, or 1.56 kw.-hr. per ton.

The experimental plant of the Inspiration Copper Co., using approximately 950 cu. ft. of air at 5-lb. pressure and treating 200 tons per day with four roughers and one half-size cleaner, required 24 hp.; deducting 4 hp. for two 2-in. centrifugal pumps the figures become 4 hp. per cell, 10 tons per horsepower, or 1.79 kw.-hr. per ton.

A maximum figure would be $2\frac{1}{2}$ kw.-hr. per ton of feed, using 5- to $5\frac{1}{2}$ -lb. air pressure, generated by Roots or Connersville positive blowers.

Costs

Oil.—The oil mixtures generally in use cost from 1.25c. per pound up to 3c. per pound, depending on the percentage of creosol and other high-priced oils used; on most ores, $1\frac{1}{2}$ c. per pound is a safe average figure, and a consumption of 1 to $1\frac{1}{2}$ lb. or from 1.25c. to 4.5c. per ton of feed, averaging $2\frac{1}{2}$ c. would be a safe estimate.

Labor.—This will vary, of course, with the size of the plant. At one plant consisting of 60 cells, two men per shift operate the entire plant, equivalent to a cost of $1\frac{1}{4}$ c. per ton. One man per shift on a 250-ton plant means a cost of 5.4c. per ton maintenance. Assuming a life of three months per blanket, a capacity of 50 tons per cell, and an allowance for repairs to blowers, motors, pumps, etc., we have $\frac{1}{2}$ c. per ton as a liberal allowance.

Power.—With power at 1c. per kilowatt-hour, and a consumption of $2\frac{1}{2}$ kw.-hr. per ton, the cost would figure 2.5c. per ton of feed.

Summarized, the estimated cost on a 2,000-ton flotation plant, or larger, would be approximately as follows:

	Per Ton, Cents
Labor	1.25
Oil	2.50
Maintenance	0.50
Power	2.50
Total	6.75

On a plant of 250 tons the extra labor costs per ton would bring it up to approximately 10c. per ton of flotation feed.

Actual figures from a large plant treating over 2,000 tons by flotation gave 6.1c. per ton; the flotation feed in this case represents 60 per cent. of the crude-ore tonnage, making the cost 3.5c. per ton of crude ore treated.

THEORIES

So far no satisfactory explanation of flotation phenomena has been advanced. At my instigation and under my direction, a large amount of research work has been done in an endeavor to formulate some logical explanation of the phenomena, and perhaps to find some scientific way of conducting flotation experiments in place of the empirical methods now in vogue. Although the latter object has not yet been attained, still these experiments have resulted in the formulation of a theory that appears to be well grounded and that may prove of interest and value to others engaged in this art.

Much work has been done at the Mellon Institute at Pittsburgh under the direction of Dr. Raymond C. Bacon, and lately by James A. Block at the station of the U. S. Bureau of Mines in Salt Lake City. The result of all of this work is summed up in the following statement:

In considering the connection between flotation phenomena and the physical properties of the minerals concerned, there are two parallelisms:

First.—It has been noticed for some time that the minerals which floated were not easily wetted by water, while those which were easily wetted did not tend to come up with the froth. This is the basis of about the only theory so far widely circulated; it was well stated by Hoover in his book, *Concentrating Ores by Flotation*.

Second.—There is a parallelism between certain electrostatic characteristics and the flotation properties of ores.

In the first-mentioned theory, surface tensions and contact angles should be considered. Certain minerals, such as galena, will float on the surface of still water, while gangue particles, since they possess a greater adhesive attraction for the water than the water's cohesive attraction for itself, will be drawn through the surface film into the interior, and sink because of their greater specific gravity. These properties of floatable minerals and gangues are increased by the presence of oil and acid. Oil sticks to galena with greater tenacity than it does to silica, and an oil surface is still less easily wetted than a galena surface. The acid in the water causes a still greater difference in the various surface tensions. This, it seems, is without question the explanation of the action in the MacQuisten process, in which the ore particles are lifted to the surface, where they can be removed by skimming off the surface layer of the liquid.

With reference to the second parallelism, it has been noticed that extremely small amounts of certain colloidal impurities, such as saponin or tannin, were detrimental to flotation; while others, such as Congo red and methylene blue, did not interfere, and were, if anything, beneficial. In classifying these, the injurious ones generally came under the head of what physical chemists call electronegative colloids, while electropositive colloids were not harmful. Suspended particles will generally migrate

when placed in an electric field, and this classification comes naturally from the direction of their migration. This migration is called electrophoresis, or electrical endosmose, and is the result of the formation of contact layers around the particles by the liquid containing them, very similar to the formation of surface films when liquids come in contact with air. These contact films almost invariably have a difference of potential between their inner and outer surfaces. An air-water contact film has, for instance, a difference of 0.055 volts, and other contact films have similar charges. This causes the particles to act like charged solids, and to be attracted by electric charges of opposite sign.

The charges on solids and non-miscible liquids can be conveniently studied on the stage of a microscope.

This work naturally led to the study of the charges exhibited by various ores and minerals, and in that work an interesting parallelism was observed; namely, that floatable minerals seemed to have positive charges, and non-floatable gangues negative charges. Some gangues were found with positive charges, but they were characteristically hard to handle, having a tendency to come up with the froth. These charges sometimes vary with the acidity or alkalinity of the liquid, and this variation is not inconsistent with the effects of acidity or alkalinity on the flotation of ores.

It has been noticed that these electrostatic properties depend on the condition of the surface of the particle and not upon the composition of the mass. For instance, lead oxide which is ordinarily negative, or neutral, when covered with a sulphide coating, takes upon itself a positive charge.

Recent investigations on the coagulation and deflocculation of slimes, on the coagulation and dispersion of colloids, and along similar lines, show that these contact film charges, although small, have an important bearing on the dispersion or coherence of particles suspended in liquid mediums. In fine suspensions and in colloidal solutions, these charges may often be neutralized by the introduction of oppositely charged ions, precipitation generally taking place whenever these charges fall below certain limits. Oppositely charged contact films have a general tendency to coalesce, while similarly charged films, if their charges are great enough to overcome natural cohesiveness, do not seem to coalesce, but to repel each other, and if the weight of the particles is small enough in relation to their size and surface, permanent dispersion will take place, the particles distributing themselves through a liquid in much the same manner that gas will fill a container.

In view of the above observations, it seems possible that flotation is the result of difference in polarity in the charges on the various ore particles, and on the bubbles. Since oil contact films and air contact films have both been proved to have negative charges, the positively charged miner-

als will adhere to either. The bubble mantles in a flotation machine are undoubtedly composed of oil, or of oil in emulsion, since pure water alone will not froth. The same forces, then, that cause oppositely charged colloids to agglomerate and precipitate, cause the minerals to adhere to the oil-covered bubbles; and the same forces that keep the particles of an oil emulsion dispersed, keep the gangue particles repelled from the bubbles.

Expressed briefly, the theory is as follows:

That oil flotation is an electrostatic process. It is a scientific fact that when a solid particle is suspended in water, the water will form around the particle a contact film which generally possesses an electric charge, the amount and polarity depending upon the nature of the surface of the particle and the electrolyte in which it is suspended. The presence of these charges can be demonstrated by the fact that the particles possessing them will migrate when placed in an electric field. It has been demonstrated that floatable particles have charges of one polarity (positive), and that non-floatable particles have charges of the opposite polarity (negative); that the froth is charged negatively and so attracts the positively charged or floatable minerals, and repels the negatively charged or non-floatable ones. It is this, it is believed, that causes the floatable minerals, galena, sphalerite, etc., to adhere to the froth, and the gangue minerals, silica, etc., to remain in the liquid where they can be discharged as tailing.

DISCUSSION

JAMES A. BLOCK, Salt Lake City, Utah.—As I have done some of the experimental work upon which this theory has been based, a brief outline of some of my experiments might not be out of place. During the last few months I have tested about 50 samples of ores, etc., the flotation characteristics of which I knew. Some of these were samples of froth and tailings from plants in actual operation, but most of them were samples taken from laboratory tests going on at the plant of the General Engineering Co., and at the station of the U. S. Bureau of Mines, Salt Lake City. In all of these experiments, no real exception to the theory was found. All of the tailing samples produced in actual flotation showed a negative polarity; all of the ores with the so-called "bad gangues," except one or two, showed some positive polarity, and these exceptions could be explained by the extreme dryness and scaliness of the gangue minerals; and all of the froth products showed positive polarity when the samples were fresh. It was often true that after standing some time, these froth samples would reverse, especially when they contained considerable oil, but this reversal can be explained if we assume that the oil forms a coating around the particles upon standing.

The question might naturally be asked, "can these electric charges

be put to any practical use?" A machine¹ used to purify clays employs very similar charges on the constituent particles of clay. This machine (see Fig. 7) consists of a revolving cylinder, made of conducting material, such as iron, about half immersed in a clay pulp in a trough. At a distance of about $\frac{1}{2}$ in. from the immersed side of the cylinder is a coarse wire screen. A direct current of about 50 volts was applied to the cylinder and the screen, the cylinder being the anode and the screen the cathode. The cylinder attracted the negatively charged kaolin, while the screen attracted the positively charged pyrite, ferric hydroxide, and aluminum hydroxide. It was found that a product running only about 17 to 20 per cent. moisture could be scraped off the upper side of the cylinder, which revolved slowly, while the impurities fell through

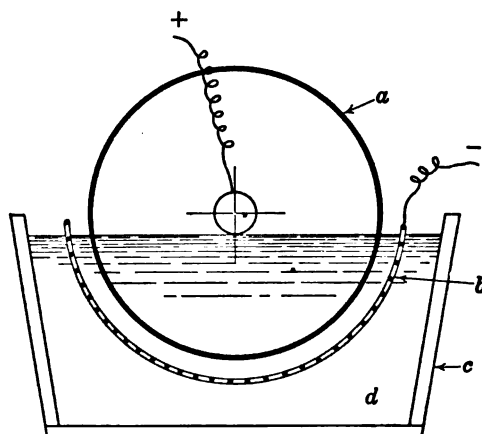


FIG. 7.—APPARATUS USED IN PURIFYING CLAYS.

- a Revolving cylinder
- b Wire screen
- c Wood trough
- d Clay pulp

the screen to the bottom of the trough. The objection to the process was the high cost of the power needed, which averaged, as I remember it, over 30c. per ton of product, but the process is undoubtedly of interest in that it shows that these electrostatic charges are dependable enough to be put to practical uses. It is also of interest in that the polarity of the charges noticed on various minerals checks the results of the experimental work upon which this theory is based.

OLIVER C. RALSTON, Salt Lake City, Utah.—Perhaps the most interesting part of Mr. Callow's paper is the reference to a theory of flotation founded on the electric charges of the ore particles, oil droplets, air bubbles, etc., involved.

¹ *Transactions of the English Ceramic Society*, vol. xii, pp. 36 to 64 (1912-13).

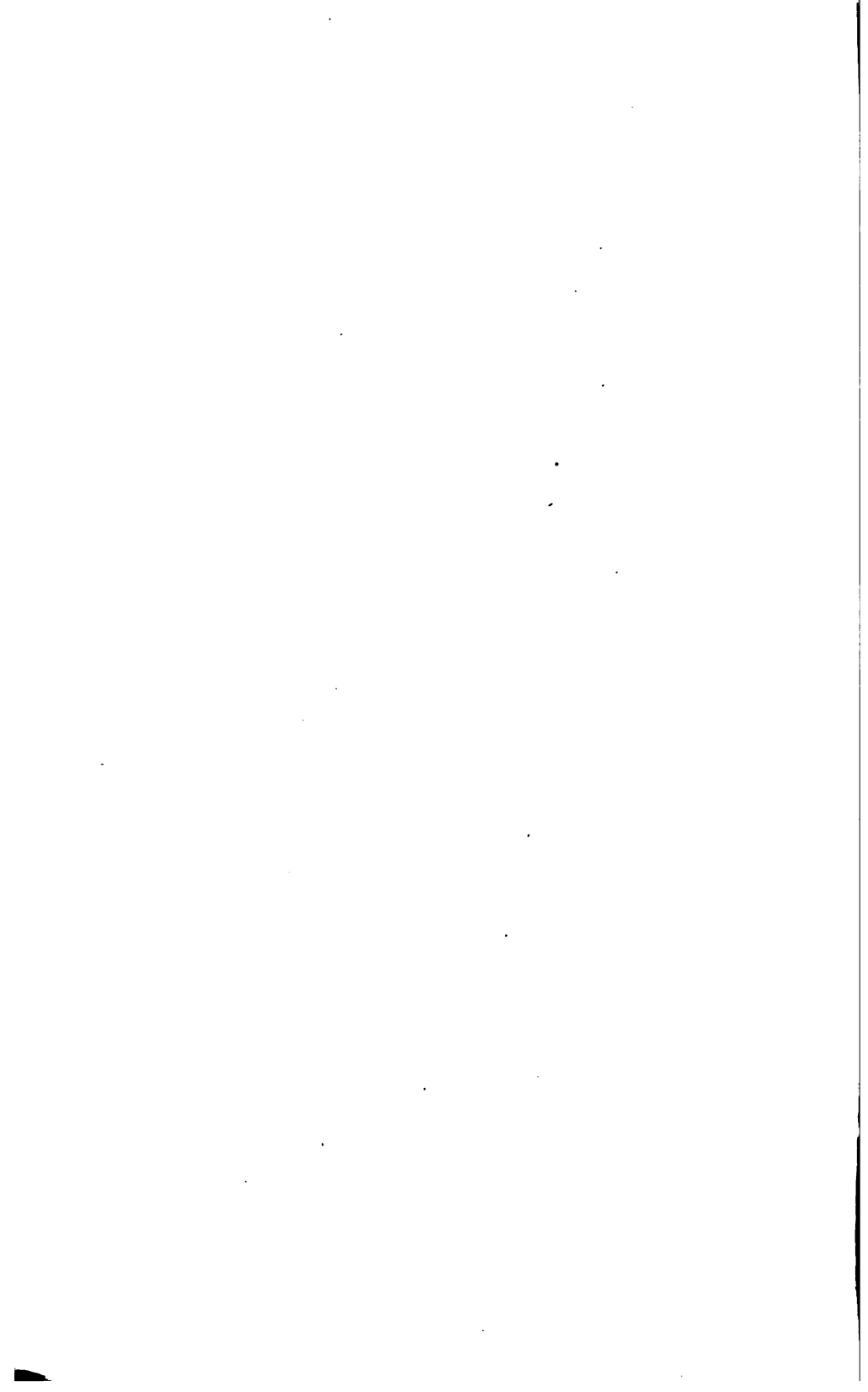
For some time I have had a growing conviction that possibly the particles dealt with in the flotation of sulphide minerals were acting according to certain laws laid down in colloid chemistry. One of the characteristic things about either suspension or emulsion colloids is that their individual particles are charged with one or the other sign of static electricity. It is known that the amount and signs of these charges can be controlled by the amount and character of electrolytes put into the water in which the colloids are dispersed, and in turn the properties of the colloidal suspension are greatly changed with this change in the electric charges on the particles. The question has been whether the comparatively coarse suspensions of ore can be compared with suspension colloids, or whether they were too coarse to allow the electric charges on the suspended particles of mineral to produce any measurable effect.

This question has been answered fairly definitely in my laboratory during the past year, for it was found that the electric charges on the particles of what we call "slimes" and "fines" are of considerable importance in the control of the rate of settling of such slimes in water. With this much definitely settled, and with a great amount of literature available in the realm of colloid chemistry dealing with the electric charges on particles of such things as quartz, galena, and sphalerite, and on oil droplets and air bubbles, it was easy to find a combination of circumstances which might explain flotation from this new viewpoint. At about this time George W. Riter made the prediction that the depths of electrostatics would have to be thoroughly sounded before flotation phenomena could be entirely understood. He seemed to have the idea at that time that the charges on the particles were generated by friction in the flotation machine, in much the same way that a piece of sealing wax becomes charged when rubbed with flannel; but we know from colloid chemistry that these charges of electricity arise whenever two substances of unlike dielectric constants come into contact.

Further work along the line of controlling the charges of the ore particles, thereby controlling the rate of settling of the particles in water, has convinced me that the charges are of considerable importance and the results of this work will be reported soon. Mr. Callow told me last spring that he had much the same conception of the matter as I, and had the Mellon Institute in Pittsburgh working on it for some time. Mr. Callow sent a man to the Salt Lake laboratory of the U. S. Bureau of Mines to try out some methods of measurement which I had proposed and which had not been used in Pittsburgh; some of the results are presented in Mr. Callow's paper. His work adds certainty to the conjecture that the electric charges on particles of minerals suspended in water are of importance in flotation phenomena. Mr. Callow has outlined one way of explaining just how these electrical effects come into play in flotation and I have outlined a second in an article in the *Mining*

and Scientific Press. The knowledge that when flotation conditions are good the froth has a positive charge and the rejected gangue a negative charge, that the oil droplets and the air bubbles are negatively charged, and the sulphide particles positively charged, is highly interesting. We have here electrical effects which are parallel to flotation phenomena, and the conclusion that they are connected with each other seems justifiable. Just exactly how they are connected is not yet clear and may prove difficult of solution.

It is to be hoped that further work will be done along this line in order that a theory of flotation may be found, which will afford a more rational control of flotation testing, and an extension of the process to the treatment of any kind of minerals.



DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

The Effect of Carbon on the Physical Properties of Heat-Treated Carbon Steel*

BY J. H. NEAD,† B. S., RACINE, WIS.

(New York Meeting, February, 1916)

OBJECT

THE experiments herein described were undertaken with a view to investigating thoroughly the influence of carbon on the tensile and impact physical properties of carbon steel. The original comprehensive plan included investigation of a series of steels with varying carbon contents over a wide range of heat treatments. These were to be studied as to their static tensile qualities, and also their resistance to shock as measured by the Charpy impact bending test. A thorough metallographic examination was likewise included in the schedule. The Charpy tests and the metallographic examination have not been completed to date, and are accordingly withheld for another report. All heat treatments, however, have been made, and the static tensile tests have been completed. A description of the procedure of the experiment and the results thus far obtained are therefore reported herein.

CONCLUSIONS

No attempt is made to draw general conclusions from these experiments, but the curves and the data are submitted as being of general importance and wide applicability.

COMPOSITION OF MATERIALS

An attempt was made to secure steels increasing in carbon by increments of 0.1 per cent. from 0.1 per cent. up to 0.8 per cent., and from that composition to 1.6 per cent. by increments of 0.2 per cent. Actually, however, steels of the composition given in Table I were procured, and as the carbon contents were arranged sufficiently well so that satisfactory curves could be drawn showing the relation between carbon content and other properties, these steels were used in the experiments.

* This work was done while the author was Metallurgical Engineer at Watertown Arsenal, Watertown, Mass.

† Metallurgical Chemist, Belle City Malleable Iron Co.

TABLE I.—*Chemical Composition of Steels Used*

Carbon, Per Cent.	Manganese, Per Cent.	Sulphur, Per Cent.	Phosphorus, Per Cent.	Silicon, Per Cent.	Chromium, Per Cent.
0.14	0.45	0.035	0.018	0.131	None
0.18	0.56	0.043	0.024	0.132	None
0.32	0.51	0.027	0.009	0.128	None
0.46	0.40	0.050	0.020	0.144	None
0.49	0.60	0.028	0.013	0.127	None
0.57	0.65	0.028	0.012	0.167	None
0.71	0.67	0.035	0.027	0.147	None
0.83	0.55	0.028	0.018	0.152	None
1.01	0.39	0.029	0.016	0.160	None
1.22	0.34	0.031	0.025	0.181	None
1.39	0.20	0.029	0.015	0.191	None
1.46	0.20	0.035	0.011	0.133	0.35

It will be noted that the carbon content varies from 0.14 to 1.46 per cent. The manganese content varies from 0.20 to 0.67 per cent., the amounts of this element being normal for commercial steels. It

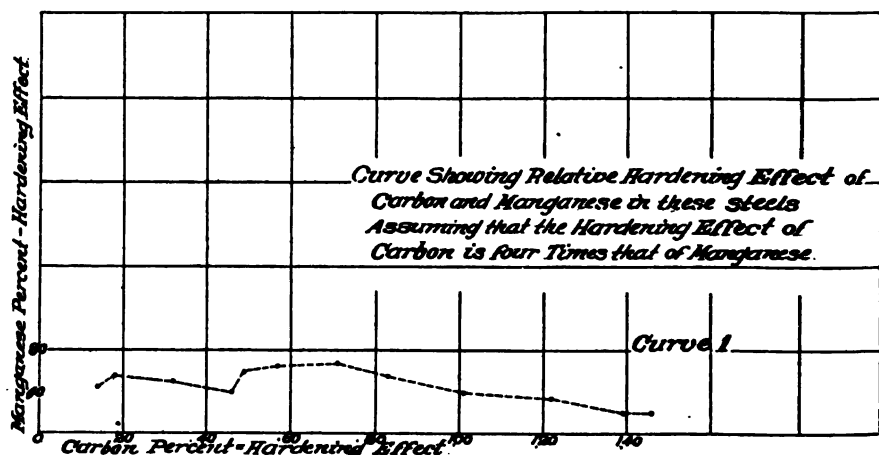


FIG. 1.

was thought that the variations in manganese content were not great enough to disturb seriously the conclusions to be drawn as to the effect of carbon.

Fig. 1 is an empirical curve showing the relative values of the hardening properties of the carbon and the manganese in these steels. The ordinate scale is reduced to one-quarter that of the abscissa scale, arbitrarily assuming that the hardening value of manganese is one-fourth that of carbon. This is an assumption often made in practice. The coördinates of any point on the curve represent, then, the relative harden-

ing effect of the carbon and the manganese in the steels whose compositions are given in Table I.

HEAT TREATMENT

No critical point determinations were made on these steels. The temperatures used in the heat treatments were based on the recommended annealing temperatures for rolled and forged carbon steel objects of the American Society for Testing Materials.¹

The recommended temperatures were plotted in the form of a curve,

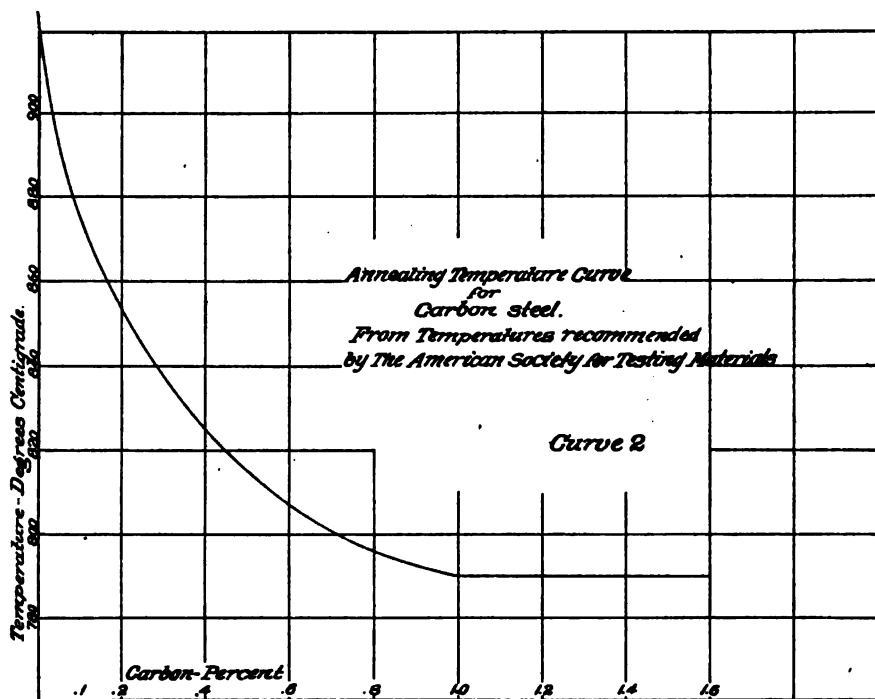


FIG. 2.

which is shown in Fig. 2. The proper heat-treating temperatures were then read from this curve.

The layout of heat treatments adopted was as follows: One tension specimen and four Charpy specimens were taken from each steel for each heat treatment. Specimens were tested: (a) As received, hot-rolled; (b) after annealing; (c) after hardening in water; (d) after hardening in oil; (e) after hardening in oil and drawing at 375°C.; (f) after hardening in oil and drawing at 460°C.; (g) after hardening in oil and drawing at 560°C.; (h) after hardening in oil and drawing at 650°C.

¹ Year Book of the American Society for Testing Materials, 1914, p. 201.

The annealing and hardening temperatures used for each steel were as follows:

Steel, Per Cent. Carbon	Annealing and Hardening Temperature, Degrees Centigrade
0.14	866
0.18	858
0.32	836
0.46	819
0.49	816
0.57	809
0.71	800
0.83	795
1.01	790
1.22	790
1.39	790
1.46	790

The steels were originally received as 12- or 14-ft. bars 1 in. in diameter. From these bars, tensile and Charpy specimens were rough-machined, and were left $\frac{1}{16}$ in. large on all dimensions. Threads were not cut on the tensile specimens, nor were the Charpy bars notched. All specimens were to be finish-machined after heat treatment. All heatings were done in a small electrically heated closed muffle furnace, which maintained a very uniform temperature throughout the heating chamber. The inside dimensions of the muffle were $3\frac{3}{4}$ by $5\frac{3}{8}$ by $11\frac{1}{2}$ in.

The heating for annealing and hardening all specimens for each steel was done at one time. The procedure was as follows: The specimens were suitably arranged in the furnace when cold. A platinum-platinum rhodium thermocouple, the hot junction of which was inserted in a sample of the steel being heated, was also arranged in the furnace. The muffle door was closed with firebrick and asbestos, and the current turned on. It required approximately $1\frac{1}{2}$ hr. for the specimens to come to heat. They were held at the desired temperature for 20 min. The specimens to be hardened were then quenched in oil or water as indicated; those to be annealed were allowed to cool slowly with the furnace, the opening to the muffle being again closed. Approximately 8 hr. was required for the furnace and annealed specimens to reach atmospheric temperature.

The drawing operation was performed as follows: A cast-iron pot, 12 by 12 by 12 in. inside dimensions, was filled with a eutectic mixture of sodium and potassium nitrates and was heated by gas. The melting point of this mixture is low enough so that the bath was molten at the desired temperatures. The bath was raised to the desired temperature and the specimens introduced in a wire basket. They were maintained at the drawing temperature for 30 min.

All temperature measurements were made by means of platinum-platinum rhodium thermocouples and a Leeds & Northrup precision

potentiometer. The thermocouples used were carefully calibrated before use.

Subsequent to heat treatment, all specimens were finish-machined

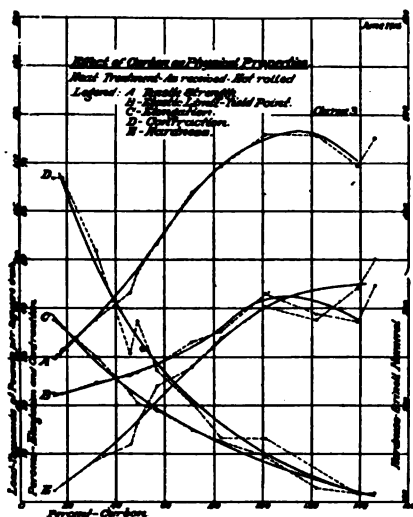


FIG. 3.

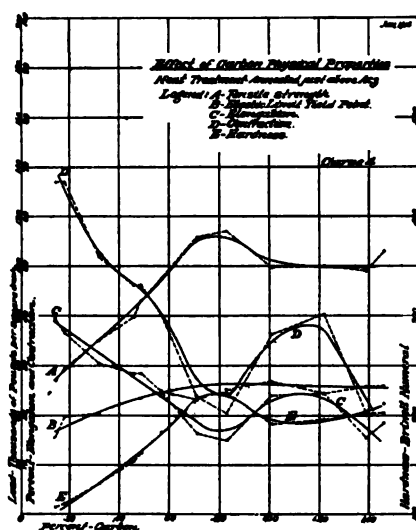


FIG. 4.

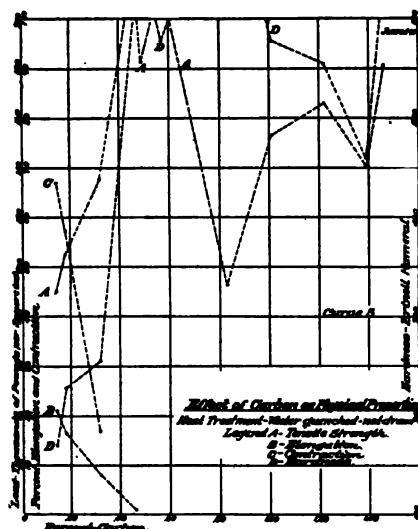


FIG. 5.

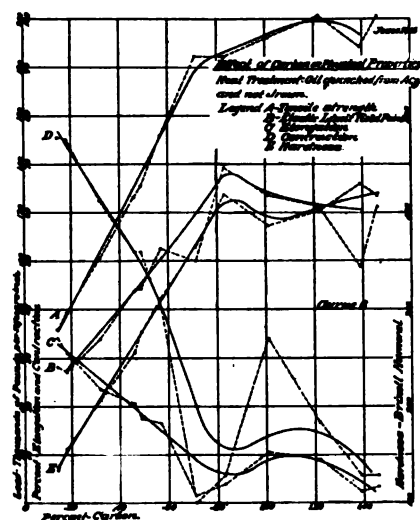


FIG. 6.

previous to testing. In the case of certain of the harder specimens, it was necessary to soften the ends so that a thread could be cut on them. This was accomplished, without affecting the reduced section of the

specimens, by partially immersing them in water and playing a flame on the end to be softened.

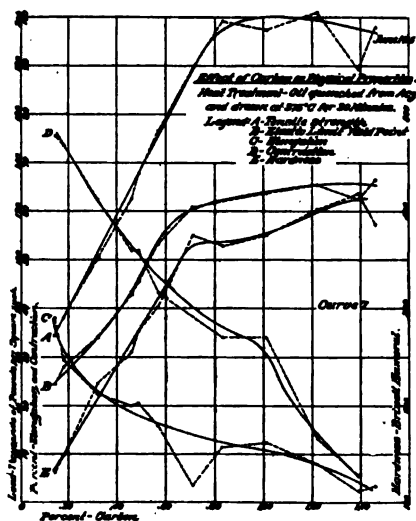


FIG. 7.

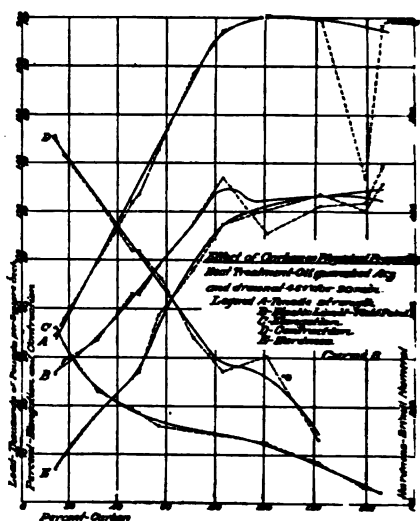


FIG. 8.

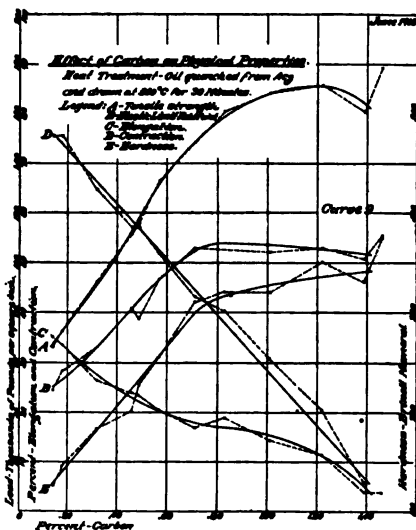


FIG. 9.

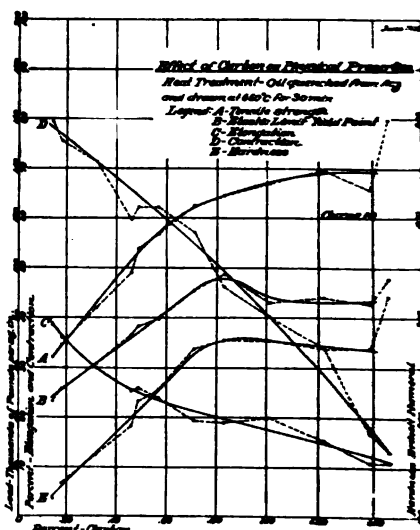


FIG. 10.

TESTING

All tensile tests were made on the 800,000-lb. Emery hydraulic-principle testing machine in the laboratory at Watertown Arsenal. Brinell hardness tests were made on an Alpha machine under standard

conditions; that is, using a 10-mm. ball, a load of 3,000 kg. and a time of application of 30 sec.

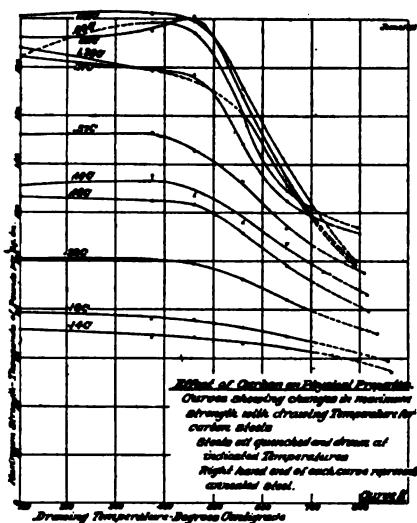


FIG. 11.

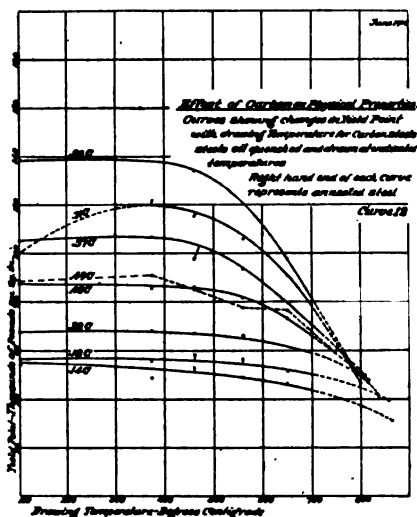


FIG. 12.

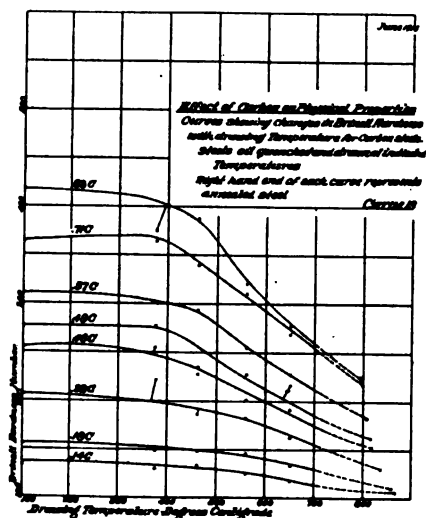


FIG. 13.

DATA

The results of the tensile and Brinell hardness tests are summarized in Tables II to IX; while Figs. 3 to 10 inclusive graphically depict these results showing for each heat treatment the changes in physical

properties due to increase in carbon content. Figs. 11, 12 and 13 show the changes in maximum strength, yield point, and Brinell hardness, respectively, with increasing drawing temperatures. Curves for the steels with different carbon content are shown on the same sheet so that

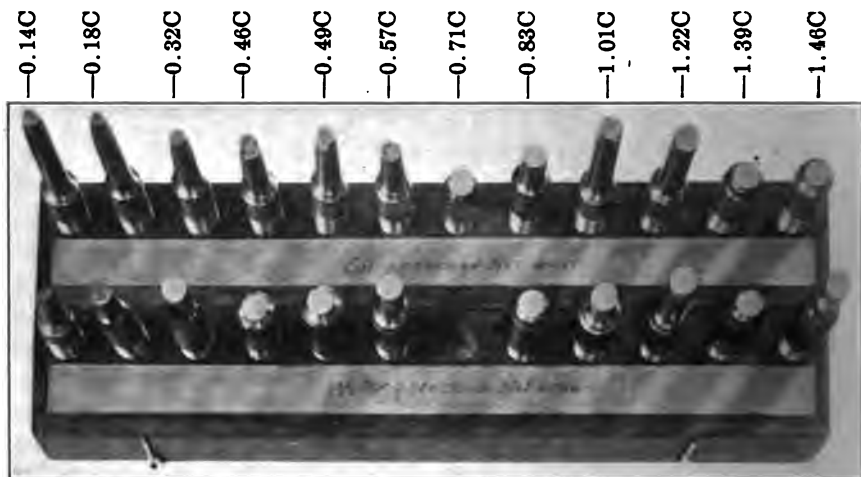


FIG. 14.

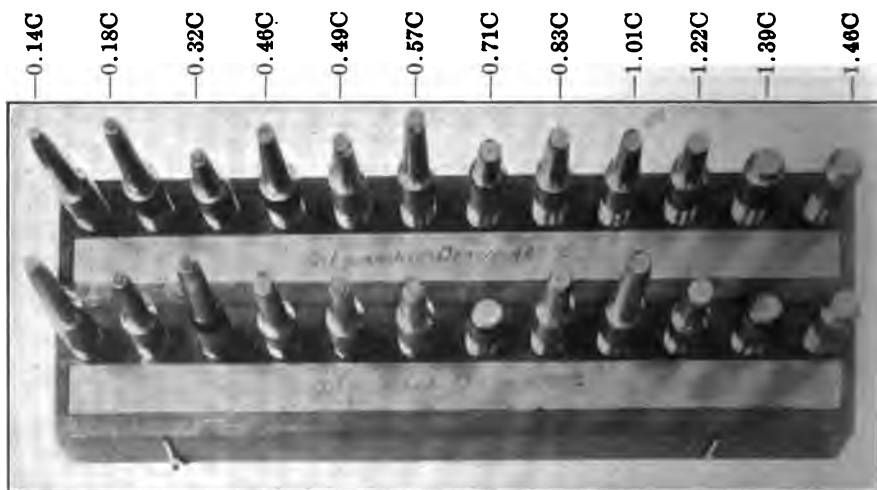


FIG. 15.

the relation between them is readily apparent. Photographs of the fractures of the tension-test specimens tested in this investigation are shown in Figs. 14 to 17. In all cases the specimens are arranged in the order of increasing carbon content from left to right, the heat treatments being indicated in the photographs.

DISCUSSION

In plotting the curves in Figs. 3 to 10 inclusive, the observed points have been located and straight dotted lines drawn between them. Idealized curves have been drawn full following the general shape of the dotted curves. Fig. 5, showing results on the water-quenched speci-

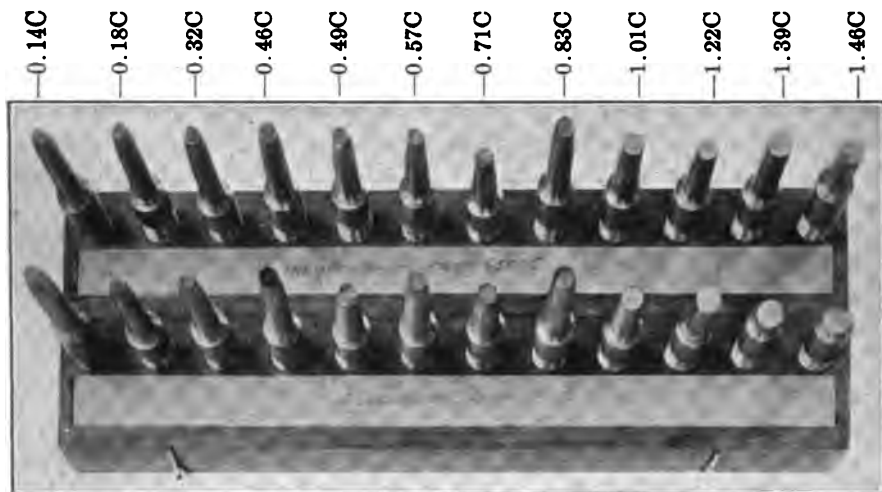


FIG. 16.

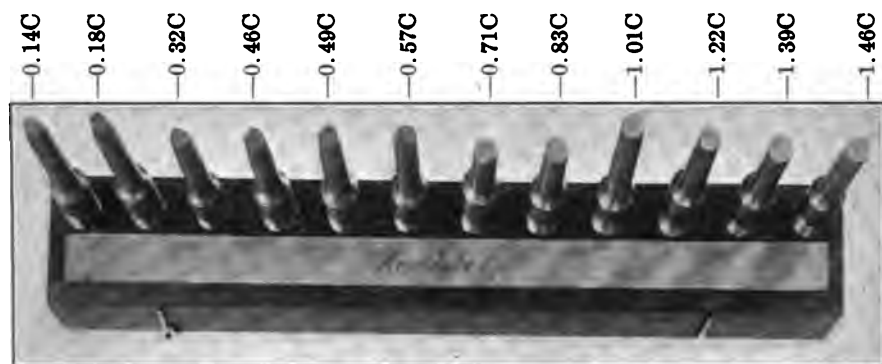


FIG. 17.

mens, is given for the sake of completeness. It is, however, of little value, for in the case of all steels examined with over 0.32 per cent. carbon, the unrelieved internal stresses doubtless caused irregular results to be obtained.

Maximum Strength.—In the annealed condition, Fig. 4, the maximum unit strength reaches a maximum value at eutectoid composition; i.e., 0.83 per cent. carbon, and then decreases slightly with increase of

carbon. With the heat-treated steels, however, Figs. 6, 7, 8, 9 and 10, and also in Fig. 3 for hot-rolled steels, the falling off in maximum strength does not begin until a composition of 1.20 per cent. carbon is reached. These curves show a rapid rise in tensile strength with increasing carbon to eutectoid composition, then a less rapid increase in strength to 1.20 per cent. carbon, followed generally by a slight falling off. The increase in strength in most cases at 1.46 per cent. carbon is to be attributed, at least partially, to the chromium content in this particular steel.

Yield Point.—In all cases, except the hot-rolled condition, Fig. 3, the unit stress of the yield point reached a maximum at eutectoid composition, falling off slightly or remaining constant with increase in carbon.

Brinell Hardness.—The curves for Brinell hardness are similar in shape in nearly all cases to the curves for maximum strength.

Elongation and Contraction.—In general, the curves representing these two qualities show decreasing values with increasing carbon. The tendency of the curves is to be concave upward—they are straighter, however, in the heat-treated steels than in the hot-rolled condition. In Figs. 4 and 6, that is with steels, in the annealed and the plain oil-quenched conditions, the curves show a gain in ductility just beyond the eutectoid composition which is unaccountable.

The curves in Figs. 11, 12 and 13 show the relation between the maximum strength, yield point, and Brinell hardness, respectively, and the drawing temperature after oil quenching for the various steels investigated. These curves show nicely the relative effects of increasing carbon in connection with hardening and quenching treatments. The final points on all the curves represent the annealed condition of the metal. The curves are, therefore, shown dotted between these last points and a drawing temperature of 700°C.

TABLE II.—*Specimens Tested, As Received*

Carbon, Per Cent.	Yield Point, Pounds per Square Inch	Ultimate Strength, Pounds per Square Inch	Elongation, Per Cent	Contraction of Area, Per Cent.	Elastic Ratio, Per Cent.	Brinell Hardness Number
0.14	45,000	59,500	37.5	67.0	75.6	112
0.18	45,000	63,000	36.0	67.0	73.0	118
0.32	49,500	75,500	30.0	51.9	65.5	144
0.46	52,500	86,500	22.5	30.7	60.7	160
0.49	54,000	95,000	20.5	37.2	56.8	183
0.57	57,000	106,500	19.0	27.4	53.5	220
0.71	66,000	128,000	15.0	20.5	51.5	240
0.83	70,500	139,000	12.5	13.3	50.7	269
1.01	86,000	152,000	9.5	13.3	56.5	302
1.22	77,500	151,500?	3.0?		51.1	288
1.39	74,500	139,000	2.0	1.8	53.6	321
1.46	89,500	150,500	1.5	1.8	59.4	351

TABLE III.—*Specimens Annealed*

Carbon, Per Cent.	Yield Point, Pounds per Square Inch	Ultimate Strength, Pounds per Square Inch	Elongation, Per Cent.	Contraction of Area, Per Cent.	Elastic Ratio, Per Cent.	Brinell Hardness Number
0.14	31,000	54,500	39.5	67.0	56.8	107
0.18	39,500	59,000	36.5	67.0	66.8	111
0.32	41,000	70,000	30.5	51.9	58.5	131
0.46	48,000	79,500	28.5	46.2	60.3	153
0.49	47,000	86,000	28.5	46.2	54.6	163
0.57	50,000	95,000	25.0	40.3	52.6	183
0.71	46,500	111,500	16.5	24.0	41.7	217
0.83	50,500	114,000	15.0	20.5	44.3	223
1.01	53,800	99,400	24.0	36.2	54.1	192
1.22	49,000	100,000	24.5	40.3	49.0	196
1.39	51,500	98,000	15.5	20.5	52.5	202
1.46	52,000	106,000	18.5	20.5	49.0	212

 TABLE IV.—*Specimens Water Quenched*

Carbon, Per Cent.	Yield Point, Pounds per Square Inch	Ultimate Strength, Pounds per Square Inch	Elongation, Per Cent.	Contraction of Area, Per Cent.	Brinell Hardness Number
0.14	Not Determined.	90,000	21.0	67.0	170
0.18		105,000	16.5	57.2	228
0.32		135,500	8.0	16.9	255
0.46		220,000	1.0	0.0	600
0.49		183,000	0.0	0.0	713
0.57		215,000	0.0	0.0	578
0.71					
0.83		93,000	0.0	0.0	744
1.01		153,000	0.0	0.0	578
1.22		166,000	0.0	1.8	555
1.39		140,500	1.0	0.0	460
1.46		181,500	0.0	0.0	627

 TABLE V.—*Specimens Oil Quenched*

Carbon, Per Cent.	Yield Point, Pounds per Square Inch	Ultimate Strength, Pounds per Square Inch	Elongation, Per Cent.	Contraction of Area, Per Cent.	Elastic Ratio, Per Cent.	Brinell Hardness Number
0.14	56,500	71,500	34.0	75.5	79.0	134
0.18	54,500	78,700	31.0	75.2	69.2	156
0.32	67,500	101,000	23.5	62.3	66.8	207
0.46	87,500	126,500	20.5	51.9	69.1	255
0.49	88,500	131,000	17.5	51.9	67.5	277
0.57	105,000	152,000	16.5	40.3	69.0	311
0.71	100,000	184,500	1.5	0.0	54.2	364
0.83	138,500	184,500	4.0	5.7	75.0	418
1.01	127,000	192,500	10.5	34.0	65.9	387
1.22	122,500	201,500	9.0	16.9	60.8	402
1.39	97,500	188,500	2.5	5.7	51.7	430
1.46	122,500	201,500	3.5	5.7	60.7	418

TABLE VI.—*Specimens Oil Quenched and Drawn at 375°C.*

Carbon, Per Cent.	Yield Point, Pounds per Square Inch	Ultimate Strength, Pounds per Square Inch	Elongation, Per Cent.	Contraction of Area, Per Cent.	Elastic Ratio, Per Cent.	Brinell Hardness Number
0.14	49,000	68,500	38.0	75.5	71.5	131
0.18	56,000	76,000	29.5	73.5	73.6	149
0.32	68,000	100,500	22.5	62.3	67.0	223
0.46	85,500	125,000	20.0	51.9	68.4	255
0.49	91,000	135,500	20.5	51.9	67.1	277
0.57	106,500	153,000	16.0	43.3	68.8	302
0.71	121,400	179,100	3.5		67.8	375
0.83		198,500	11.5	34.0		364
1.01	127,500	195,000	12.5	34.0	65.3	375
1.22	131,000	203,000	8.0	13.3	64.5	402
1.39	125,000	178,500	2.5	5.7	70.0	418
1.46	133,000	196,000	3.5		68.8	387

TABLE VII.—*Specimens Oil Quenched and Drawn at 460°C.*

Carbon, Per Cent.	Yield Point, Pounds per Square Inch	Ultimate Strength, Pounds per Square Inch	Elongation, Per Cent.	Contraction of Area, Per Cent.	Elastic Ratio, Per Cent.	Brinell Hardness Number
0.14	53,000	68,500	36.0	75.5	77.3	134
0.18	58,500	75,500	32.5	71.4	77.4	149
0.32	67,000	99,000	23.0	62.3	67.6	187
0.46	86,000	123,500	19.5	51.9	69.6	228
0.49	85,500	127,000	18.5	51.9	67.3	235
0.57	97,500	145,500	16.0	46.2	64.4	293
0.71	115,500	177,000	10.0	34.0	65.2	340
0.83	134,000	194,500	14.0	37.2	68.8	387
1.01	111,000	201,500	12.5	40.3	55.0	402
1.22	122,500	199,000	8.5	13.3	61.5	418
1.39	123,500	134,000	3.5		91.4	402
1.46	139,000	197,000	2.5		70.5	430

TABLE VIII.—*Specimens Oil Quenched and Drawn at 560°C.*

Carbon, Per Cent.	Yield Point, Pounds per Square Inch	Ultimate Strength, Pounds per Square Inch	Elongation, Per Cent.	Contraction of Area, Per Cent.	Elastic Ratio, Per Cent.	Brinell Hardness Number
0.14	49,000	66,000	35.0	75.5	74.2	126
0.18	56,500	72,500	33.5	75.5	77.9	146
0.32	66,000	92,500	26.5	64.7	71.3	183
0.46	81,500	111,500	24.0	57.2	73.0	202
0.49	77,500	116,000	23.5	57.2	66.8	228
0.57	93,500	133,000	20.5	51.9	70.3	255
0.71	106,000	148,500	17.0	43.3	71.3	311
0.83		161,000	19.0	40.3		321
1.01	104,500	168,500	14.5	30.7	62.0	321
1.22	106,000	171,500	11.5	20.5	61.8	351
1.39	101,500	161,000	4.0	5.7	65.5	332
1.46	111,000	178,500	4.0		62.2	375

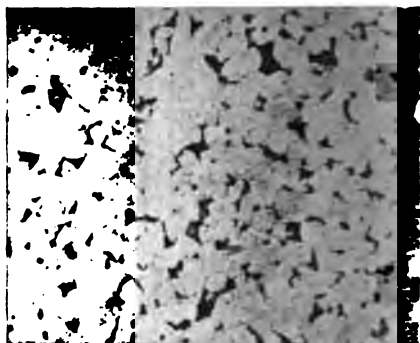
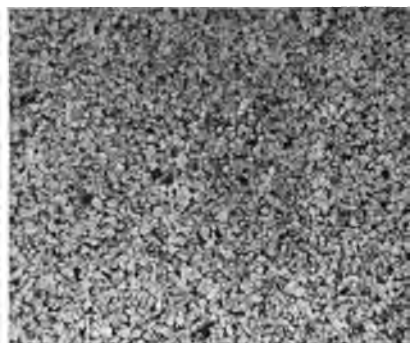
TABLE IX.—*Specimens Oil Quenched and Drawn at 650°C.*

Carbon, Per Cent.	Yield Point, Pounds per Square Inch	Ultimate Strength, Pounds per Square Inch	Elongation, Per Cent.	Contraction of Area, Per Cent.	Elastic Ratio, Per Cent.	Brinell Hardness Number
0.14	46,500	64,000	38.5	79.5	72.6	118
0.18	51,500	69,000	36.0	75.5	74.6	134
0.32	61,500	84,000	30.0	71.4	73.2	163
0.46	73,000	98,000	25.5	59.8	74.4	192
0.49	76,500	107,500	26.0	62.3	71.1	217
0.57	79,500	113,500	24.0	62.3	70.0	228
0.71	91,000	125,000	19.5	57.2	72.7	269
0.83	97,500	129,000	19.0	46.2	75.5	277
1.01	86,000	134,000	20.0	40.3	64.1	277
1.22	88,000	139,500	15.5	34.0	63.1	269
1.39	85,500	131,500	11.0	16.9	65.0	269
1.46	95,500	159,500	11.0	13.3	59.2	321

APPENDIX

The accompanying microphotographs (Figs. 18 to 29) are presented as being of considerable interest in connection with the preceding experiments. The 12 steels investigated are shown in two different conditions: (1) as received, hot-rolled and (2) normalized or annealed from 1,000°C. In the normalized series the effect of carbon on the microstructure is most readily apparent. The micrographs of the hot-rolled steels appear on the left-hand side of the illustrations; those of the normalized steels on the right-hand side.

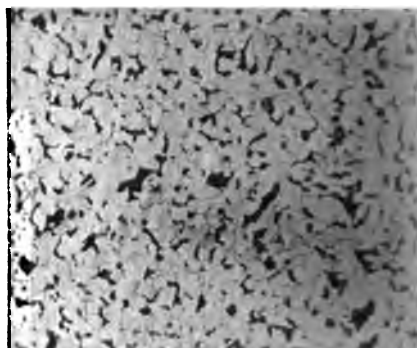
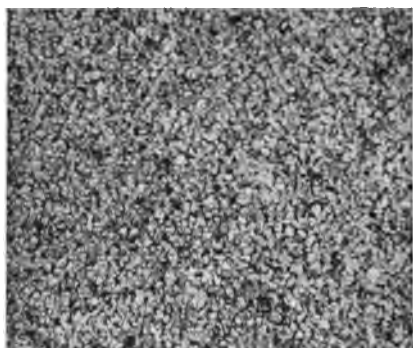
In the case of the hot-rolled steels, the structures consist of sorbito-pearlite and ferrite for the hypo-eutectoid steels, and sorbito-pearlite and cementite for the hyper-eutectoid steels. In the case of the normalized steels, the grain structures are considerably larger and the constituents are pearlite and ferrite for hypo-eutectoid steels, and pearlite and cementite for hyper-eutectoid steels. All micrographs are magnified 51 diameters.



As received. Hot rolled. Brinell hardness, 112.

Normalized. Annealed from 1,000°C. Brinell hardness, 107.

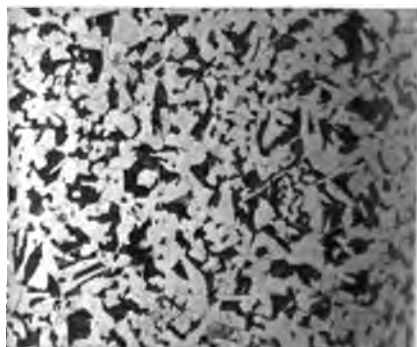
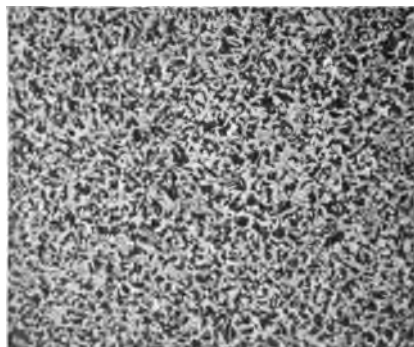
FIG. 18.—ANALYSIS: C, 0.14; Mn, 0.45; Si, 0.131; S, 0.035; P, 0.018 Per Cent.



As received. Hot rolled. Brinell hardness, 118.

Normalized. Annealed from 1,000°C. Brinell hardness, 111.

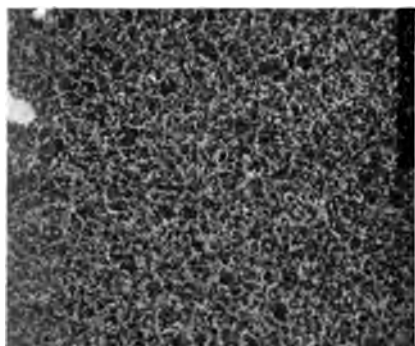
FIG. 19.—ANALYSIS: C, 0.18; Mn, 0.56; Si, 0.132; S, 0.043; P, 0.024 Per Cent.



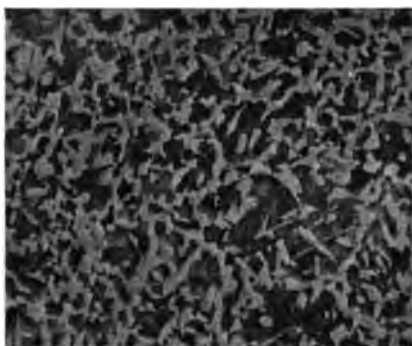
As received. Hot rolled. Brinell hardness, 144.

Normalized. Annealed from 1,000°C. Brinell hardness, 134.

FIG. 20.—ANALYSIS: C, 0.32; Mn, 0.51; Si, 0.128; S, 0.027; P, 0.009 Per Cent.

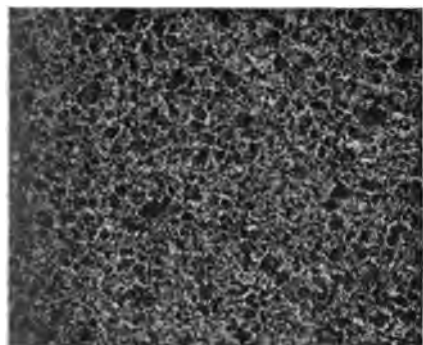


As received. Hot rolled. Brinell hardness, 160.

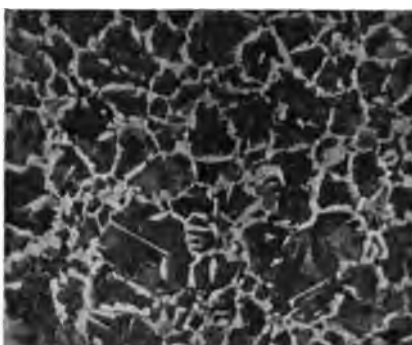


Normalized. Annealed from 1,000°C. Brinell hardness, 155.

FIG. 21.—ANALYSIS: C, 0.46; Mn, 0.40; Si, 0.144; S, 0.50; P, 0.020 Per Cent.

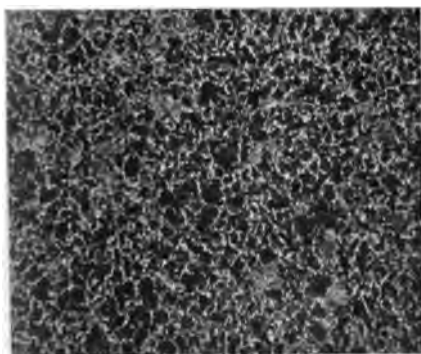


As received. Hot rolled. Brinell hardness, 183.

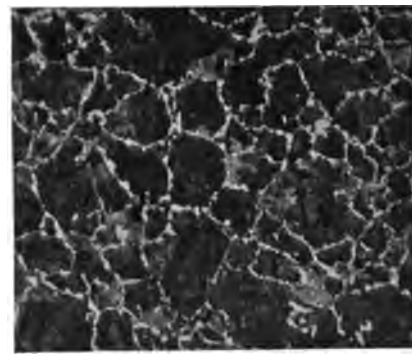


Normalized. Annealed from 1,000°C. Brinell hardness, 179.

FIG. 22.—ANALYSIS: C, 0.49; Mn, 0.60; Si, 0.127; S, 0.028; P, 0.013 Per Cent.



As received. Hot rolled. Brinell hardness, 220.



Normalized. Annealed from 1,000°C. Brinell hardness, 206.

FIG. 23.—ANALYSIS: C, 0.57; Mn, 0.65; Si, 0.167; S, 0.028; P, 0.012 Per Cent.



As received. Hot rolled. Brinell hardness, 240.

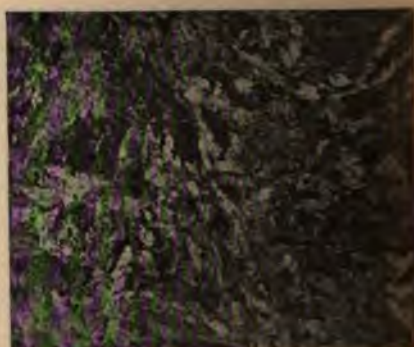


Normalized. Annealed from 1,000°C. Brinell hardness, 235.

FIG. 24.—ANALYSIS: C, 0.71; Mn, 0.67; Si, 0.147; S, 0.035; P, 0.027 Per Cent.

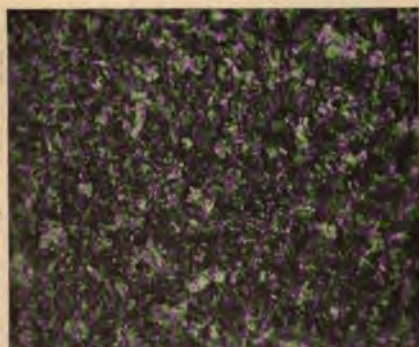


As received. Hot rolled. Brinell hardness, 269.



Normalized. Annealed from 1,000°C. Brinell hardness, 257.

FIG. 25.—ANALYSIS: C, 0.83; Mn, 0.55; Si, 0.152; S, 0.028; P, 0.018 Per Cent.



As received. Hot rolled. Brinell hardness, 302.



Normalized. Annealed from 1,000°C. Brinell hardness, 255.

FIG. 26.—ANALYSIS: C, 1.01; Mn, 0.39; Si, 0.160; S, 0.029; P, 0.016 Per Cent.

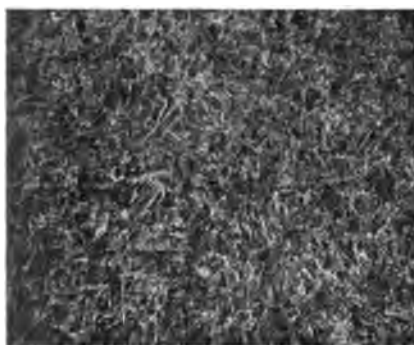


As received. Hot rolled. Brinell hardness, 288.

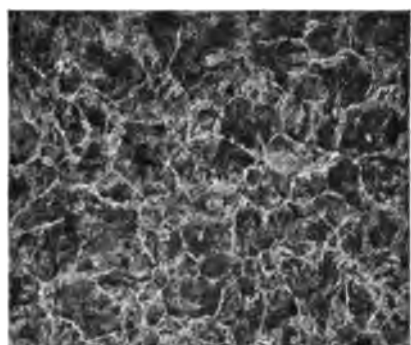


Normalized. Annealed from 1,000°C. Brinell hardness, 269.

FIG. 27.—ANALYSIS: C, 1.22; Mn, 0.34; Si, 0.181; S, 0.031; P, 0.025 Per Cent.



As received. Hot rolled. Brinell hardness, 321.

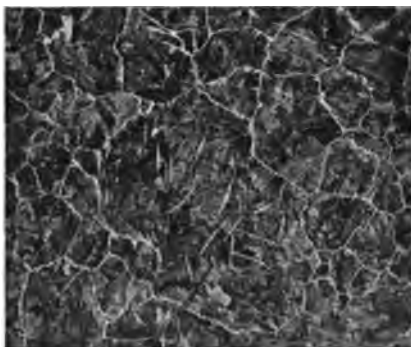


Normalized. Annealed from 1,000°C. Brinell hardness, 255.

FIG. 28.—ANALYSIS: C, 1.39; Mn, 0.20; Si, 0.191; S, 0.029; P, 0.015 Per Cent.

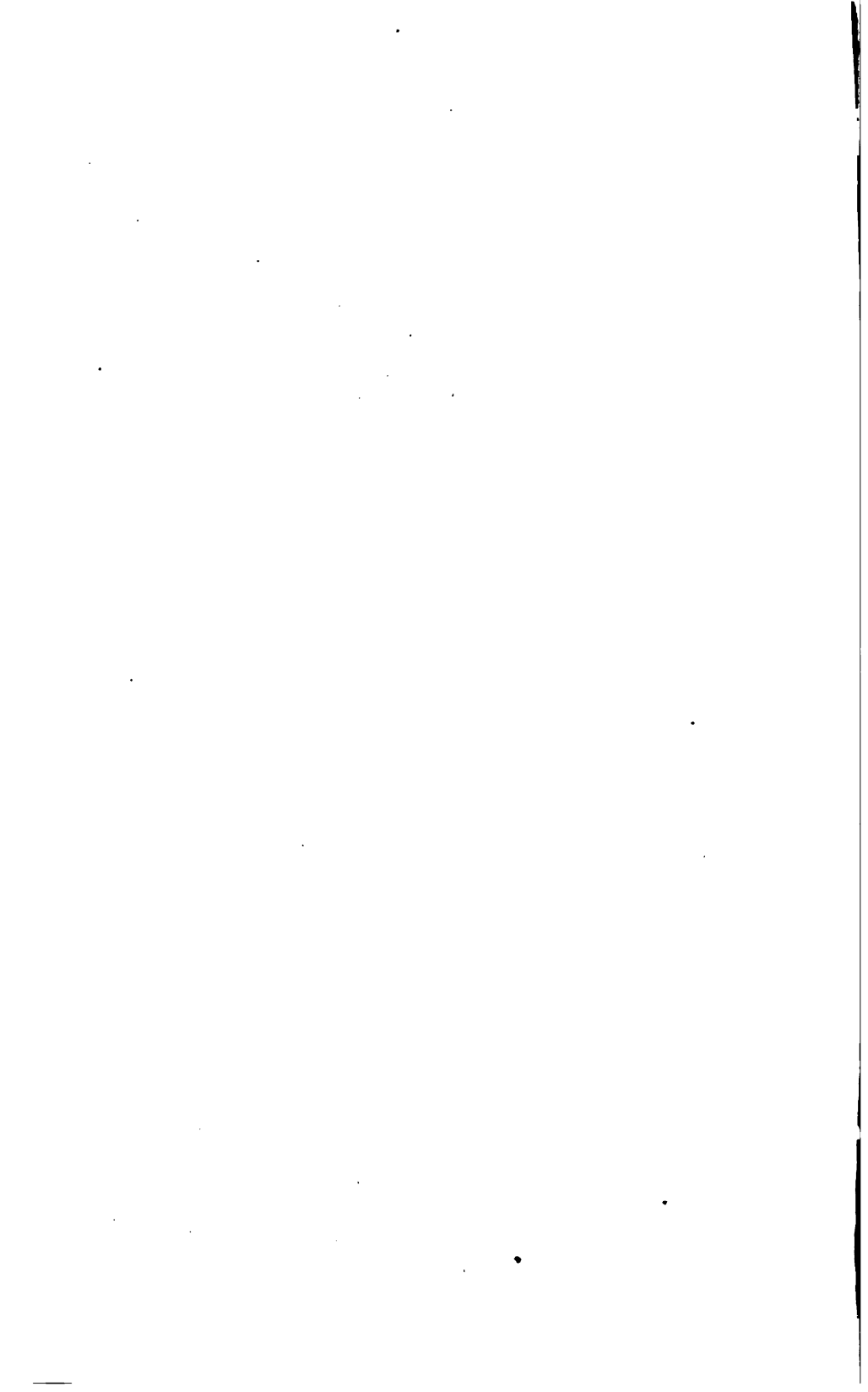


As received. Hot rolled. Brinell hardness, 351.



Normalized. Annealed from 1,000°C. Brinell hardness, 302.

FIG. 29.—ANALYSIS: C, 1.46; Mn, 0.20; Si, 0.133; S, 0.035; Cr, 0.35; P, 0.011 Per Cent.



DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

The Determination of Grain Size in Metals*

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(New York Meeting, February, 1916)

It is well known that many properties of a given metal vary with the size of grain or cell. For most industrial purposes, where high ultimate strength and high elastic limit are desired, the manufacturer tries to produce a fine-grained structure. For some purposes, for example in transformer iron, a coarse-grained structure is desirable.

The terms fine-grained and coarse-grained are used only in a relative sense. For example, a fine-grained cast copper might have grains 100 times larger than coarse-grained high-speed steel. Similarly, a fine-grained steel rail might have grains or cells 100 times larger than coarse-grained wire, which was made of steel from the same heat.

In some metals, the change in grain size is more appreciable than the changes in any of the properties determined by the tensile test. Furthermore, in metals which have been annealed or subjected to high temperatures, grain size may prove to be a better indication of the life of the metal in use than the tensile test. One of the authors was able to supplant the tensile test with grain-size determinations in the control of metal for a specific use, the latter determination being more indicative of the life of the metal part than the tensile test. A grain-size determination with a tensile test is, of course, more valuable than either singly.

At present there is a decided dearth of data on actual grain-size determinations in metals, and consequently a corresponding lack of ability to interpret the determinations after they are made. It would be desirable to have correlation made between grain size and properties of metals, comparing thousands of determinations. In this way, grain-size determinations might be interpreted in much the same manner that tensile tests are now.

* A contribution from the metallurgical laboratory of Case School of Applied Science, Cleveland, Ohio.

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SEVERAL METHODS OF MEASURING GRAIN SIZE

One of the chief reasons why more grain-size measurements have not been made is the excessive work involved in making a determination. The planimeter method is accurate—in fact, too accurate for the nature of the work involved—but it is slow and tedious. It involves the tracing of the outside boundary lines of a group of grains, counting the grains, measuring the area with a planimeter, and a calculation which is different for each determination. Furthermore, the method requires a planimeter, which piece of apparatus might not be available in all metallographic laboratories.

We have developed a method by means of which a grain-size determination can be made in one-fifth the time required by the planimeter method. The method is as accurate as the sampling of the specimen and does not require apparatus which is not available in metallographic laboratories.

In this paper it is proposed: (1) to outline the more important methods now used for the determination of grain size in metals, (2) to outline a new method developed by us, and (3) to compare the various methods.

The Planimeter Method

This method may be used in either of two ways. The image of the metal specimen may be projected on to a piece of paper and the grain

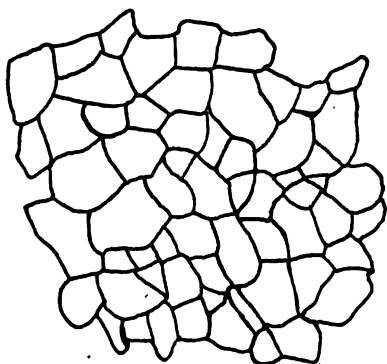


FIG. 1.—PLANIMETER METHOD.

boundaries traced, as shown in Fig. 1. The total area divided by the number of grains inclosed will give the average area per grain. The second way, which involves less time, is to trace on the paper just the outside lines bounding the area of a group of whole grains. The number of included grains is then indicated by check marks on the paper, as shown in Fig. 2. Since the counting of the grains can be done at the

time of checking, this way is the shorter. The following example is taken from an actual determination:

The area found by measuring with a planimeter was 5.92 sq. in. The number of grains inclosed was 61. Therefore, the average area per grain is $\frac{5.92}{61} = 0.09705$ sq. in. The observation was made at a magnification of 100 diameters. Then $\frac{0.09705}{100 \times 100} = 0.000009705$ sq. in., which is the actual area per grain.

$$\frac{1}{0.000009705} = 103,040 \text{ grains per square inch.}$$

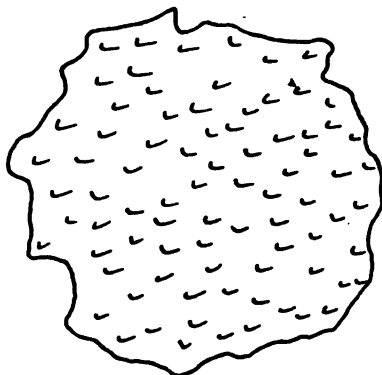


FIG. 2.—PLANIMETER METHOD, MODIFIED.

Heyn's Method

The Heyn method¹ depends for its accuracy on the assumption that the intercepts of a straight line intersecting a number of grains will be proportional to the square roots of the areas of the grains. In other words, the square of the average intercept will be the average area per grain. Heyn applies this assumption as shown in Fig. 3. The number of grains is counted along a line such as *AB*. The first incomplete grain is counted as one and the last grain is not counted. By taking a number of lines parallel to *AB* and counting the grains intersected by each, the average intercept in one direction is obtained. The average intercept in the direction *CD*, at right angles to *AB*, is obtained in a similar way. The product of the two average intercepts is the average area per grain. For example, suppose the number of intercepts in the direction *AB* is 79 and the distance 20 in. The average intercept is $\frac{20}{79} = 0.2533$ in. Likewise, suppose the number of intercepts counted

¹ Reports of the Technische Hochschule zu Charlottenburg, *The Metallographist*, vol. vi, pp. 54 to 63 (1903).

in the direction CD is 72 and the distance is 16 in. The average intercept in this direction is $\frac{16}{72} = 0.222$ in. Then the average area per grain is $0.222 \times 0.2533 = 0.05623$ sq. in. If the magnification is 100 diameters, then $\frac{0.05623}{100 \times 100} = 0.000005623$ sq. in., actual area per grain, or $\frac{1}{0.000005623} = 177,800$ grains per square inch.

It is not necessary to trace the grain boundaries to apply Heyn's method. A piece of cross-section paper may be used for a screen and the grains intersecting each line counted and the distance measured.

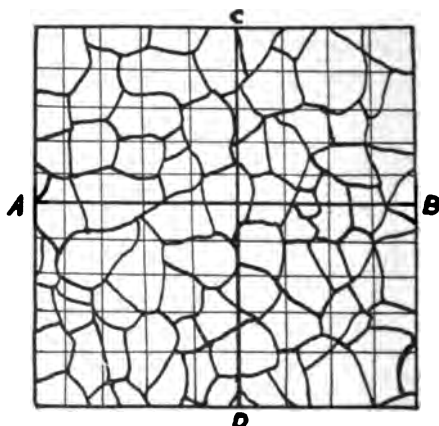


FIG. 3.—HEYN METHOD.

The Intercept Method

The intercept method is based on the same assumption as Heyn's method, but the readings are not confined to series of parallel lines. The method of procedure for determining the grain size by the intercept method is as follows: In Fig. 4, the line AB is drawn through the center of the field. This line intersects 13 grains in a length of 2.58 in. The average intercept per grain is therefore $\frac{2.58}{13} = 0.198$ in. $(0.198)^2 = 0.0392$ sq. in., the average area per grain. If we consider the magnification as 100 diameters, then $\frac{0.0392}{100 \times 100} = 0.00000392$ sq. in., the actual area per grain. The number of grains per square inch is $\frac{1}{0.00000392} = 255,000$. In order to get the average diameter per grain, sufficient lines should be taken to intersect from 50 to 500 grains.

- In actual practice, this method is shortened considerably.² An

² H. M. Howe; Life History of Network and Ferrite Grains in Carbon Steel. *Proceedings of the American Society for Testing Materials*, vol. xi, pp. 380 to 381 (1911).

eye-piece with a graduated scale may be used in place of the regular eye-piece. The length of an even number of grains can be measured direct and the approximate grain size obtained in a short time.

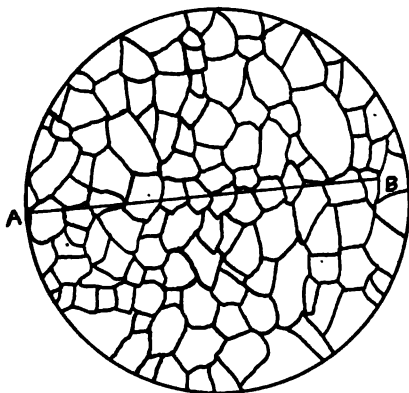


FIG. 4.—INTERCEPT METHOD.

Authors' Method

Our method has for its object the quick determination of the equivalent number of whole grains included within a circular portion of an image. It is evident that some of the grains will be completely included within

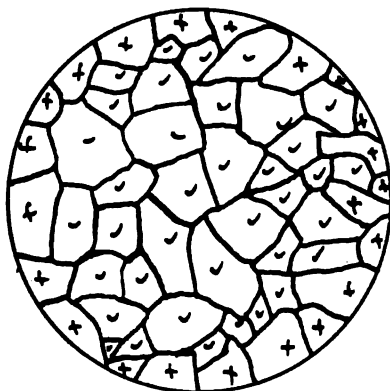


FIG. 5.—AUTHORS' METHOD.

and some will be intersected by the circumference of the circle. The former will have no correction factor. The boundary grains, however, will be partly inside and partly outside the circle. In Fig. 5, the grains marked with a check mark are completely inclosed and those marked + are the boundary grains. The average portion of each boundary grain included within the circle may be found as follows: The grains completely

included within the circle are first counted and checked. Then, the boundary grains are checked with a different mark and counted. (If the counting is done at the time of checking, the time of making a determination will be shortened.) In Fig. 5, there are 37 included grains and 22 boundary grains. The area of the included grains by planimeter is 1.9 sq. in. The area of the circle (diameter 1.83 in.) is 2.635 sq. in.

$$1.9:2.635 = 37:X$$

$X = 51.3$ the equivalent number of whole grains within the circle.

$51.3 - 37 = 14.3$ the equivalent number of whole grains in the inside portions of the 22 boundary grains. $\frac{14.3}{22} = 0.65$ average portion of each boundary grain within the circle.

If this factor be determined on a large number of samples, it may be used on unknown samples to find the equivalent number of whole grains within a circular area. If we call this factor y , the total number of grains within a circle is,

$$z + yw,$$

where z is the number of grains completely inclosed and w , the number of boundary grains.

The factor y has been determined empirically on 175 samples. The results are given in Table I. The first column represents the number of grains completely included within the circle.

TABLE I.

Grains Completely Included	Average Value of y	Number of Determinations
10-20	0.556	11
20-30	0.582	15
30-40	0.542	13
40-50	0.574	10
50-60	0.590	11
60-70	0.600	12
70-80	0.576	11
80-90	0.577	10
90-100	0.596	10
100-120	0.551	11
120-140	0.618	11
140-160	0.546	10
160-180	0.586	10
180-200	0.558	11
200-230	0.606	10
230-290	0.622	9

The average value of the factor y for 175 determinations is 0.581. The same factor can be used for all numbers of grains included within the observed area, but it is recommended that all important determinations have at least 50 grains included within the circle. Since 0.581 is nearer to 0.6 than to 0.5 we have adopted 0.6 as a general factor for all determinations. It should be noted that an error of 1 per cent. in the determination

of the whole-grain equivalent of the boundary grains makes an error of less than 0.25 per cent. in the final result, since more than 75 per cent. of the total area of the circle is occupied by the included grains.

The following is an example of how the factor is used:

Inclosed grains = 66.0

Boundary grains = 28.0

Factor = 0.6

$66 + 0.6 \times 28 = 82.8$ grains within the circle.

From the magnification, the number of grains per unit of area can be readily calculated.

COMPARISON OF THE VARIOUS METHODS WITH THE PLANIMETER METHOD

In making the calculations for the comparison of methods, the actual number of grains per unit of area was not figured, the number of grains per square inch on the drawing being sufficient for comparison. The percentage of error was figured in the following manner. Suppose the planimeter method gave 50 grains per square inch, the intercept method 54.2, Heyn's method 55.6. The percentage of error in the intercept method is $\frac{4.2}{50} \times 100 = 8.4$ per cent.; and in Heyn's method $\frac{5.6}{50} \times 100 = 11.2$ per cent. Tables II, III, and IV show the comparative results obtained by using the different methods.

TABLE II.—*Heyn's Method*

[illegible]

TABLE III.—*Intercept Method*

Total Length of Line, Inches	Total Number of Grains Cut	Average Diameter of Grain	Area of Grain	Area of Grain by Planimeter	Planimeter Method, Grains per Square Inch	Intercept Method, Grains per Square Inch	Error, Per Cent
16.70	87	0.1920	0.0369	0.0425	23.5	27.1	15.30
20.75	121	0.2275	0.0540	0.0353	28.4	19.5	31.30
17.80	98	0.1820	0.0330	0.0380	26.4	30.4	15.20
19.63	105	0.1780	0.0317	0.0412	24.3	31.5	28.90
18.50	107	0.1725	0.0297	0.0352	28.4	33.6	18.20
20.20	112	0.1810	0.0325	0.0357	28.0	30.8	10.00
20.45	115	0.1780	0.0317	0.0387	25.9	31.5	21.50
17.60	99	0.1780	0.0317	0.0438	22.9	31.5	37.50
20.30	119	0.1703	0.0291	0.0334	30.0	34.4	14.80
31.40	114	0.2760	0.0763	0.0875	11.4	13.1	14.90
34.70	135	0.2570	0.0660	0.0815	12.3	15.1	23.20
30.50	144	0.2120	0.0450	0.0529	19.0	22.2	17.10
32.52	120	0.2720	0.0740	0.0773	12.8	13.6	4.60
32.06	139	0.2310	0.0531	0.0810	12.4	18.8	51.50
33.58	135	0.2490	0.0620	0.0641	15.6	16.2	3.82
33.88	139	0.2450	0.0596	0.0682	14.6	16.2	15.00
33.88	138	0.2460	0.0605	0.0885	11.3	16.5	50.20
Average percentage of error.....							21.90

TABLE IV.—*Authors' Method*

Included Grains	Boundary Grains	Factor	Area of Circle	Authors' Method, Grains per Square Inch	Planimeter Method, Grains per Square Inch	Error, Per Cent
118	49	0.6	7.068	147.4	146.30	0.8
95	35	0.6	7.068	116.0	123.80	6.3
34	27	0.6	7.068	50.2	49.85	0.7
59	30	0.6	7.068	77.0	76.22	1.0
74	37	0.6	7.068	96.2	94.93	1.3
56	35	0.6	7.068	77.0	73.69	4.5
110	41	0.6	7.068	134.6	137.80	2.3
93	40	0.6	7.068	117.0	116.97	0.0
75	37	0.6	7.068	97.2	94.20	3.2
103	43	0.6	7.068	128.8	134.00	3.9
Average percentage of error.....						2.4

In Heyn's method and the intercept method, the results are persistently high. In the former, all of the errors are plus except one, and that is minus only 0.41 per cent. In the intercept method, only one minus error is found.

In our method, the average of plus and minus errors is minus 0.1 per cent., which, of course, is less than the accuracy of the determinations.

QUICK METHOD OF GRAIN-SIZE MEASUREMENT BY AUTHORS' METHOD

A quick way to make a grain-size determination by our method has been developed. A circle 79.8 mm. in diameter is used. In making a grain-size determination, one of the magnifications given in Table V is used. In the third column is given the factor, for each magnification, by which the equivalent number of whole grains within the circle is to be multiplied to obtain the number of grains per square millimeter.

Magnification Used	Diameter of Circle, Millimeters	Factor
25	79.8	$\frac{1}{4}$
50	79.8	$\frac{1}{2}$
100	79.8	2.0
150	79.8	4.5
200	79.8	8.0
250	79.8	12.5
500	79.8	50.0
750	79.8	112.5
1,000	79.8	200.0
1,500	79.8	450.0
2,000	79.8	800.0

The following example shows how this table is used: Included grains, 61; boundary grains, 35; magnification, 100. Grains per square millimeter = $(61 + 0.6 \times 35)2 = 164$.

Similar tables can be made for square-inch calculations, or for circles of different diameters.

DISCUSSION

Our results show that the intercept method and Heyn's method are not accurate.

The largest error found in 187 determinations by our method was 6.4 per cent., the average error 2.1 per cent., and the average algebraic deviation from the planimeter measurements, plus 0.25 per cent.

An average of 30 representative determinations showed that 78.5 per cent. of the total area of the circle was occupied by the included grains. This portion is determined accurately by our method. The factor is used in determining only 21.5 per cent. of the number of grains. An error of 5 per cent. in estimating the boundary-grain portion makes an error of slightly over 1 per cent. in the final result.

A square or any other shape might be used instead of a circle. It seems probable, if a square or rectangle were used, that the factor would be 0.5 instead of 0.6, when the circle is used.

SUGGESTED RULES FOR SAMPLING

Some metallographic specimens have more than 5,000,000 grains exposed on one face. It becomes necessary to obtain a representative

sample of this face. We have tentatively adopted these general rules which, of course, can be changed as occasion demands.

1. Use such a magnification that about 50 grains will be included within the circle. (It is better to have more than 50 grains included within the circle than less, but it is always recommended that an even number of magnifications be used, even though the number of included grains should exceed 100.)

2. Make at least five determinations at about equal intervals along the diameter of the specimen. The average of all of the determinations is taken as the final result. It is evident that if there is much difference in grain size between the edge and center of the specimen, the former should have the greater weight. Such a case, however, may come under rule 4.

3. If the samples are small enough (for example, small wire), it is advisable to determine the total number of grains on transverse sections, and the total number for a given length on longitudinal sections. It is sometimes practical to do this with larger coarse-grained specimens, either macroscopically or at low magnifications.

4. When one part of a specimen differs greatly from another part in grain size, do not average the separate determinations, but record each and note the position of each determination by means of a free-hand sketch.

5. In samples which have been cold worked, the relative degree of cold work in any direction is expressed by the ratio of the length divided by the width of the grains. For obtaining this ratio, it is recommended that Heyn's method be used.

APPARATUS AND MANIPULATION FOR GRAIN-SIZE MEASUREMENTS

Any good metallurgical microscope with camera attachment can be used for grain-size determinations. It is recommended that only even magnifications be used. For instance, if an observation is being made at a magnification of 469 diameters, the setting should be changed to 500 diameters.

Grain-size measurements can be made on a screen or on a photograph. Either a ground glass, or a piece of paper fastened to a clear glass, may be used as a screen. A convenient system, where records of the determinations are kept, is to cut several pieces of thin typewriter paper so that they fit the clear-glass screen, and draw a circle with the desired diameter (we use 79.8 mm.) near the center of each sheet. Fasten one of these pieces of paper to the clear-glass screen by means of gummed labels, or, otherwise, and project on to it the image of the specimen.

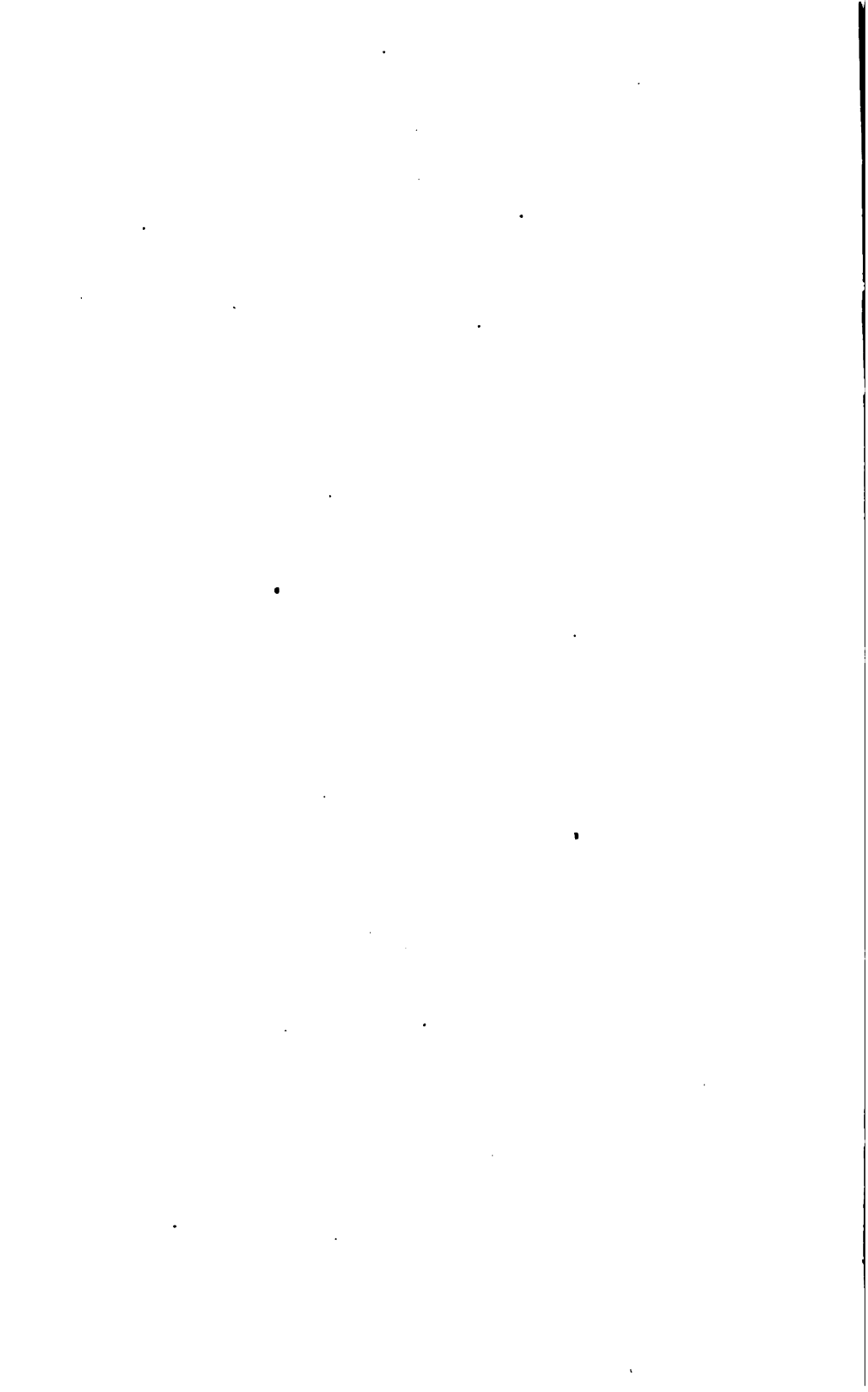
The circumference of the circle should be well within the image. Count and check separately the boundary grains and the included grains,

make calculation for grains per unit area and save the sheet of paper with check marks and calculations. This will serve as a permanent record of the determination.

If the existence of a grain boundary is doubtful, often a slight change of focus will remove the doubt. It is, therefore, essential that the fine focusing screw be provided with an extension so the focus can be changed while the operator examines the image on the screen.

Unless the room is rather dark, or the microscope lamp excessively strong, the operator will find it helpful to work under a cloth hood.

The specimen should be etched so as to bring out distinctly the grain or cell boundaries.



DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Magnetic Studies of Mechanical Deformation in Certain Ferromagnetic Metals and Alloys

BY H. HANEMANN AND PAUL D. MERICA,* B. A., PH. D., WASHINGTON, D. C.

(New York Meeting, February, 1916)

THE application of other than mechanical methods to the study of the mechanical-physical properties of metals has become in the last few years a topic of investigation of ever-increasing interest, both from the theoretical standpoint and from that of the engineer and constructor. Besides the investigations which have been made of the magnetic properties of ferromagnetic metals, in their relation to the mechanical properties of these metals, which will be discussed more in detail below, there have been made studies of other phenomena in the same relation, and methods for the mechanical testing of metals have even been suggested, for example, the microscopic method of Tammann,¹ and the thermo-electric method of Turner² for determining elastic limits.

The application of all such methods, which are based upon the dependence of some physical, crystallographic, or possibly chemical property, upon the mechanical state of the metal, must be preceded by close and careful correlation of these various "test" properties with the known mechanical state of the different metals or materials. It is with this view that the present work, as well as that of many previous investigators of, what may be termed in the interests of brevity, the mechanical-magnetic properties of ferromagnetic materials of construction, has been done.

PREVIOUS INVESTIGATIONS OF MECHANICAL MAGNETIC PROPERTIES

The effect of mechanical stresses and of deformation, elastic or otherwise, on the magnetic qualities of iron, steel, and other metals, has been studied by a number of investigators from the time of Joule (1841) to the present. It will, however, for the present purpose be sufficient to give an account only of the principal results obtained in the course of these investigations.

It was early noticed (Villari, 1865) that if an iron wire is placed in a

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¹ *Zeitschrift für Elektrochemie*, vol. xviii, p. 584 (1912).

² *Proceedings of the American Society of Civil Engineers*, vol. xlviii, p. 140 (1902).

magnetic field of low value, the values of the magnetic induction, measured while the wire is subjected to tension, will first increase with the tension, reach a maximum value and afterward decrease. This phenomenon is called the "Villari reversal," after the discoverer, and was later studied and verified by Sir William Thompson (1877 to 1879).

In a general investigation, covering in its scope the study of the magnetic characteristics of iron and steel and the effect of temperature and of mechanical stress upon these characteristics, Ewing made the first systematic contribution to our knowledge of the mechanical-magnetic properties of iron. Ewing worked with iron and steel wires; he studied the effect of tensile stresses upon the form of the B - H curve (the curve giving the relation between the magnetizing force, H , and the total induction, B); he noticed that iron which had already been stretched beyond the elastic limit (yield point) was more sensitive magnetically to stress than before, and showed that a permanent deformation or set resulted in a permanent decrease in the magnetic permeability. He also studied rather minutely certain magnetic-mechanical hysteresis effects, which will be referred to later in the present paper.

In 1902, Fraichet in a short investigation presented a new aspect of the matter, namely, the applicability of a magnetic method to the study of the mechanical properties of iron and steel. Fraichet placed his specimens in a testing machine, applied a constant magnetizing force, and proceeded to carry out the ordinary tensile test on the specimen, through which the change of magnetic flux or induction was continually indicated, but not measured, by a galvanometer in circuit with a secondary coil surrounding the specimen. He obtained in this way two curves, one, the ordinary stress-strain diagram, and the other, a curve indicating the changes taking place in the permeability of the material of the specimen as a function of the tensile stress to which it was subjected. He observed that the yield point and elastic limit were indicated on the latter curve by wide fluctuations of the galvanometer reading.

Smith and Sherman (1914) have made measurements, using the Burrows compensation method, of the permeability of certain steel specimens during the tensile test, with a view to determining the effect of tensile stress on the form of the B - H curve. Their results confirm those of the older investigations, as well as our own, in showing the Villari effect.

Goerens (1912 and 1913), in a comprehensive study of the effect of cold working on the various physical properties of iron and steel, has included studies on the magnetic properties. He found that the effect of cold working was to diminish the permeability, which was restored, however, upon annealing.

Burrows (1912 to 1914), in a number of articles, discusses the correla-

tion of mechanical and magnetic properties, and suggests using the measurements of the latter to determine the former.

THE AUTHORS' INVESTIGATIONS

The work, of which a partial account will be given here, was done in 1912 and 1913 at the Königlische Technische Hochschule in Charlottenburg-Berlin.^{*} It was considered that the important work of the previous investigators had not solved, or in some cases even considered, some questions which are of vital interest from a practical standpoint; for example, the question as to the magnetic definition of elastic limit, or in other words, the determination of the magnetic change which takes place at that value of the stress called the true elastic limit.

Another question, apparently left unsettled by previous investigation, was the effect of variations of the mechanical stress below the true elastic limit upon the magnetic properties for zero load, or in other words, the question as to whether the magnetic properties of iron and steel are dependent upon their previous elastic history. The value of such a relation, if it existed, is obvious, for it would then be possible, in the investigation of "failed" or other ferrous material, to determine by the aid of magnetic measurements what were the values of the stresses to which the specimen had been previously subjected.

The investigations of Ewing mentioned above have conclusively shown that, provided an iron or steel specimen is not brought to what may be called its normal state by a process of demagnetization before each magnetic measurement, the values of any magnetic quantity so measured are dependent upon the previous mechanical history, even if the stresses applied were below the elastic limit. The utilization in any practical way of the value of a magnetic quantity measured without previous demagnetization is, however, extremely limited, for the reason that there is lag, or hysteresis, in the relation of other variables than stress to permeability; this is true, for example, with magnetizing force. It has been found, however, that the effects of this hysteresis or lag can, in the latter case, be removed by demagnetization, to such an extent that the permeability is a determinable function of the field strength. The question then that remains is whether a stress application within the elastic limit affects the magnetic quantities measured after removal of the stress and demagnetization.

It was intended, further, to give particular attention to the discovery of a possible relation of the position of the permeability maxima in the permeability-stress curves to the values of the mechanical constants, such as yield point and elastic limit.

^{*} A full account of the investigation was published in January, 1914, as a dissertation of the University of Berlin.

So far, the investigation has been limited to the study of the magnetic properties as affected by mechanical tension, and has consisted in simultaneous measurements of magnetic and mechanical quantities of the materials during the tensile test. The magnetic measurements were carried out on the specimen in the testing machine.

Description of Materials and Methods Used

To avoid complications as much as possible, materials were chosen for the investigation which were homogeneous in structure and thoroughly annealed at the outset. The annealing insured that the material was free from initial stress of any sort which would tend to obscure the results obtained. That this annealing is necessary is shown in a comparison of results obtained on material as received and as afterward annealed.

The principal material used was a soft iron of low carbon; a wrought iron, a sample of nickel, and a nickel steel were also used. The analyses of these metals are shown in Table I.

TABLE I.—*Metals Used in Tests*

Number	Material	Analysis, Per Cent.
1	Soft iron.....	C, 0.08; Si, 0.04; P, 0.008; Mn, 0.23; S, 0.026.
2	Wrought iron.....	
3	Nickel.....	Fe, 1.70; Co, 0.77.
4	Nickel steel.....	C, 0.45; Ni, 20.40.

These materials were furnished in the form of rods, which were turned down to the test shape desired, 50 to 55 cm. in length and from 0.8 to 1.20 cm. in diameter, and then annealed in an atmosphere of nitrogen, or *in vacuo*, at a temperature of 950°C. This was done in a long electric furnace specially constructed for this work. The specimens were allowed to cool in the furnace, unless otherwise noted.

In outline, the measurements were conducted as follows: The test bar, inserted through a specially constructed magnetizing coil, was placed in the testing machine. An extensometer was attached, which measured the elongation immediately adjacent to the part of the test bar included within the magnetizing coil. Stresses of various values were imposed, and magnetic and extensometer measurements taken.

The Magnetic Measurements.—Only a brief description of the apparatus used in the measurements of the magnetic quantities is given here. For further details reference should be made to the dissertation previously mentioned. The total inductions, B , were measured ballistically in the usual manner; the values of the field strength, by the aid of the mag-

netic-potential-meter (*Spannungsmesser*) described by Rogowski and Steinhaus.⁴

In order to determine extremely small changes in the magnetic induction, the latter was not measured directly, but in each case the difference between the permeability of the specimen under stress and that of a similar specimen of the same material, outside of the testing machine and under no stress, was measured. In this manner it was possible to measure changes in the induction of from 0.2 to 1.0 gauss,

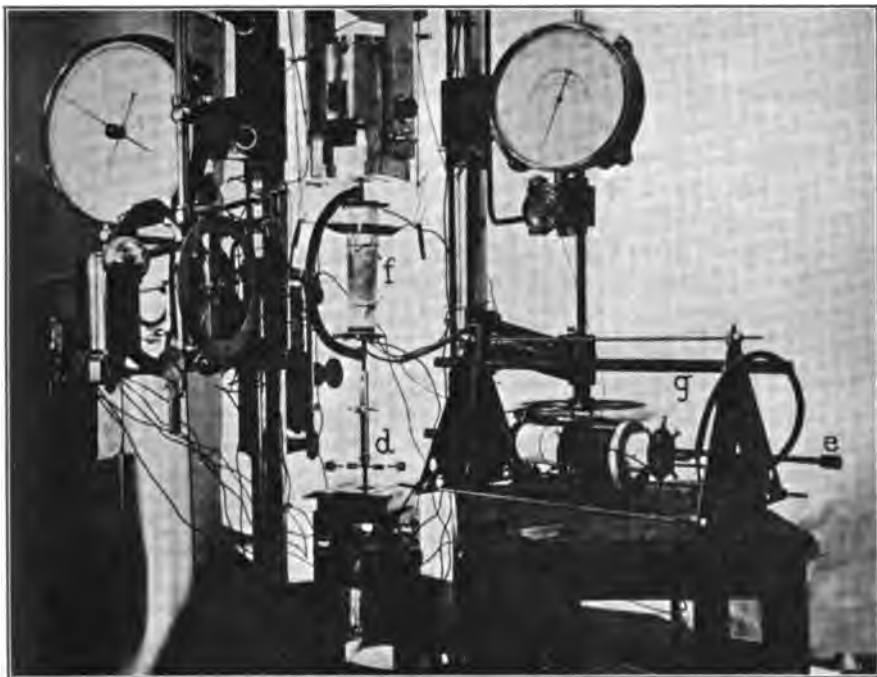


FIG. 1.—TENSILE-TESTING MACHINE EQUIPPED TO MAKE SIMULTANEOUS MAGNETIC AND MECHANICAL MEASUREMENTS

depending upon the value of the total induction. The measurement of the absolute value of the permeability itself was probably subject to a maximum error of about 2 to 3 per cent.; that of the variations of the permeability with applied tension, to a less error.

Physical Measurements.—The physical tests were made in a 5,000-kg. Losenhausen universal testing machine. The elastic elongations were measured with the well-known Marten's mirror extensometer. The elastic limit will be defined here as that least value of the stress at which there remained a permanent elongation of 0.0002 per cent., since variations, in the test length of 15 cm., of this amount could be measured,

⁴ Mitt. Phys. Tech. Reichsanstalt (1912).

and successive determinations of the length at zero load agreed to within this amount.

Fig. 1 shows the whole apparatus as set up in the testing machine ready for a series of measurements. The test specimen is shown in the jaws of the testing machine, with the spool *f*, containing the magnetizing and secondary coils, suspended independently, and with the mirror extensometer *d* attached. The compensating bar *e* is shown in the yoke *g*.

At the beginning of a series of measurements, the specimen in the machine was demagnetized and the *B-H* curve directly determined for zero load. Thereupon the magnetizing and secondary coils of the comparison bar *e* were thrown into circuit in such a way that the ballistic throw of the latter was in the opposite direction to that of the secondary of the test bar in the machine, almost wholly neutralizing it when the test bar was not stressed. Both bars were now demagnetized by the same current, the magnetizing force removed, and a stress applied to the test bar. Both bars were again demagnetized and the difference between the values of the total induction in the stressed and in the unstressed comparison bar directly determined, for various values of the field strength, in the usual way. This process was always adhered to; any change in the stress applied to the test bar was made only after demagnetization and removal of the magnetizing force, and was always followed by demagnetization of both bars. The temperature of the bars was kept constant during the measurements by a cooling coil, with running water, which was made a part of the magnetizing spool *f*.

Results and Discussion

Types of the curve series obtained are shown in Figs. 2 and 3. These show the total induction *B* for constant field strengths as a function of the applied tensile stress (abscissas) for the soft iron (material 1) as annealed, and as annealed and elongated 5.7 per cent. in the testing machine, respectively. The curves have maxima, which occur at large values of the stress at low values of the magnetizing force *H*, and lesser values of the stress with increasing values of *H*. For higher values of *H*, these maxima disappear entirely.

The positions of these maxima, that is, the values of the stress at which they occur in each case, are different in the annealed and in the stretched or elongated material. The maxima occur in the elongated material at higher values of the stress than in the case of the annealed material before it has undergone any plastic deformation. It is well known that certain mechanical constants occur also at higher values of the stress in cold worked or stretched than in soft or annealed material, i.e., the elastic limit, the yield point, and the ultimate strength. The

question presents itself as to what relation there is, if any, between the displacements of these magnetic maxima in the induction and the displacements of these latter mechanical constants, as iron is cold stretched or elongated.

In order to study this question, several series of measurements were made, of which the results are shown in Fig. 4. A test specimen was

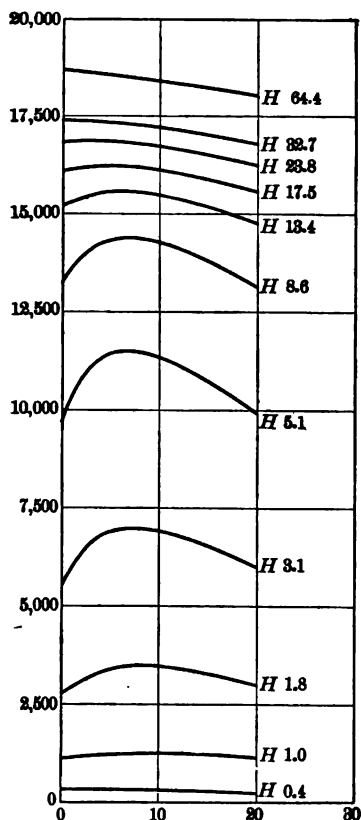


FIG. 2.—MATERIAL 1—IRON, ANNEALED. ABSCISSAS—STRESSES IN KILOGRAMS PER SQUARE MILLIMETER. ORDINATES—INDUCTIONS IN GAUSSSES.

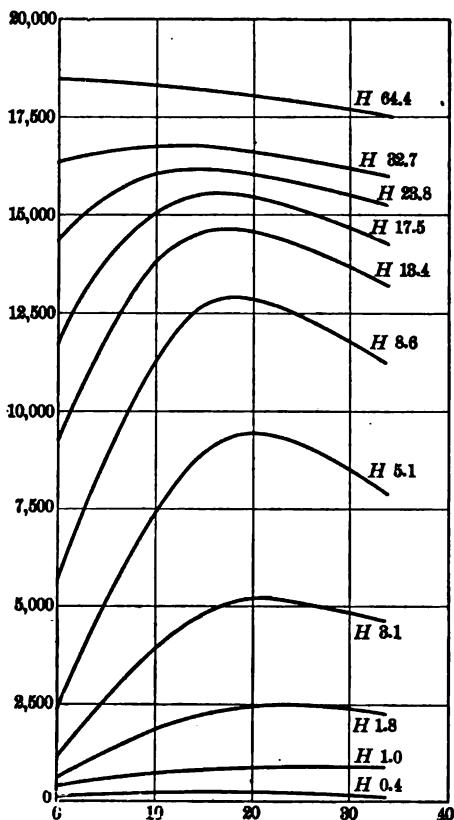


FIG. 3.—MATERIAL 1—IRON, ANNEALED AND 5.7 PER CENT. ELONGATED.

loaded up to and beyond the yield point, unloaded, loaded again up to and beyond the new yield point, unloaded, reloaded, etc. Each time the specimen was loaded beyond the yield point it was elongated permanently by about 2 per cent. The values of the induction B for four values of the field strength and the physical constants (elastic limit and yield point) were determined throughout the series of measurements. The total elongation of 5.8 per cent. was made in three steps, viz., 2.1,

3.7, and 5.8 per cent. After each of these deformations, the induction curve of the material for each value of H followed a different curve, giving with the original, or virgin curve, four in all; these are shown in Fig. 4.

The elastic limit of this material was 14.0 kg. per square millimeter, and the yield point, 20 kg. per square millimeter in the annealed condition. The induction curve for the field strength 3.4, for example,

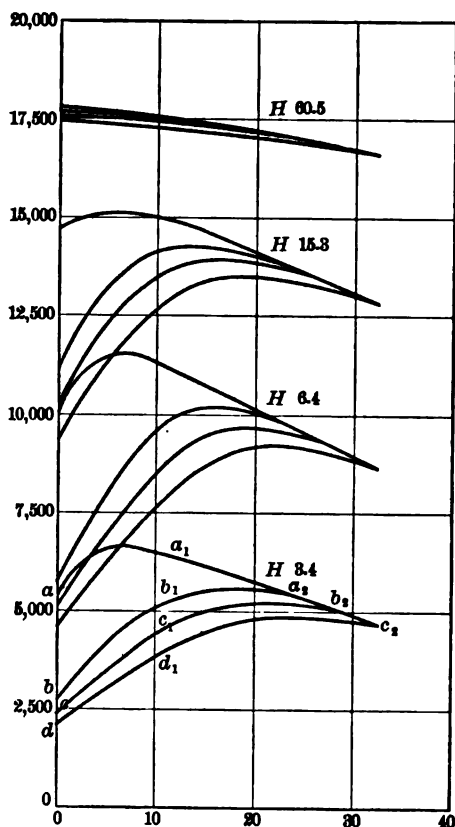


FIG. 4.—MATERIAL 1—IRON, PREVIOUSLY ANNEALED. ABSCISSAS—STRESSES IN KILOGRAMS PER SQUARE MILLIMETER. ORDINATES—INDUCTIONS IN GAUSSSES.

followed reversibly the curve aa_1 up to the elastic limit, and beyond the yield point to a_2 , at which point the specimen had been elongated by 2.1 per cent. Upon unloading, the curve a_2b_1b was followed as well as upon reloading. The stress was then increased to the value b_2 , whereby the specimen was further elongated by 1.6 per cent. Upon unloading, the new curve b_2c_1c was followed. The load was again applied (curve $cc_1b_2c_2$) and removed after a further elongation of 2.1 per cent.; the induction followed the curve c_2d_1d . The material was therefore tested

in four "states," i.e., after different degrees of homogeneous cold working, after which both the mechanical and the magnetic constants were different.

It can be seen from the curves that as the material is stretched, and the elastic limit and the yield point are raised, the positions of the induction maxima are also raised, for all values of the field strength. In Fig. 5 is shown the relation between the elastic limit and the positions of the induction maxima, for the four values of the field strength. That these are closely related is more strikingly shown in Fig. 6, in which are plotted as abscissas the values of the elongations undergone (a measure of the cold work put on the specimen), and as ordinates, the quotient of the position of the induction maxima (value of the stress at which the

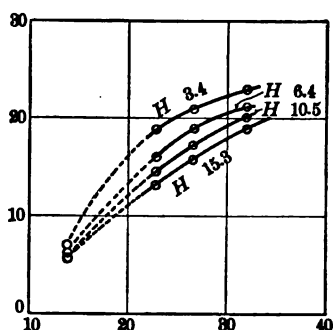


FIG. 5.

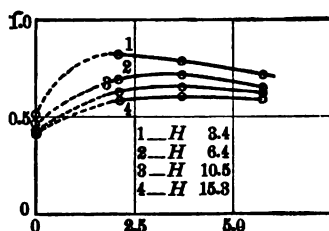


FIG. 6.

FIG. 5.—MATERIAL 1—IRON, ANNEALED. ABSCISSAS—ELASTIC LIMITS IN KILOGRAMS PER SQUARE MILLIMETER. ORDINATES—POSITION OF THE MAXIMA IN THE INDUCTION IN KILOGRAMS PER SQUARE MILLIMETER.

FIG. 6.—MATERIAL 1—IRON, ANNEALED. ABSCISSAS—ELONGATION PER CENT. ORDINATES—QUOTIENT OF THE POSITION OF THE INDUCTION MAXIMA DIVIDED BY THE ELASTIC LIMIT.

maxima occurs) divided by the value of the elastic limit. Although this quotient is not a constant, it is nearly so after the material has received an initial elongation. The quotient for these field strengths is about 0.45 for the annealed material, and is raised to 0.6, and above, by the first elongation of 2.1 per cent., remaining thereafter approximately constant.

It is interesting also to notice that, whereas there is a large change in the permeability under zero stress after elongation, there is practically none during elongation under the stresses producing it, the curve of inductions and stresses showing no discontinuity, or even marked change of direction, at the yield point.

Careful measurements were made to determine the exact minimum value of the stress at which a permanent change in the induction under zero load occurred, after removal of the load, and to relate this to the elastic limit (the minimum value of the stress at which a permanent

change in the length occurs). A specimen was subjected to cyclical loading, between the limits A and zero, until the test length included by the extensometer remained constant to within less than 0.0002 per cent. A , the upper stress limit, was increased by steps of 2 kg. per square millimeter until a permanent set was obtained. The values of the induction for two values of the field strength were also measured differentially each time after removal of the load. The results of these measurements are shown in Fig. 7, in which are plotted as abscissas the stress, and as ordinates the values of the induction B for the field strengths 34.5 and 5.2. The elastic limit lay, as nearly as could be determined, at 14 kg. per square millimeter, and the yield point at 23 kg. per square millimeter. It can be seen from the curves, that the permanent change

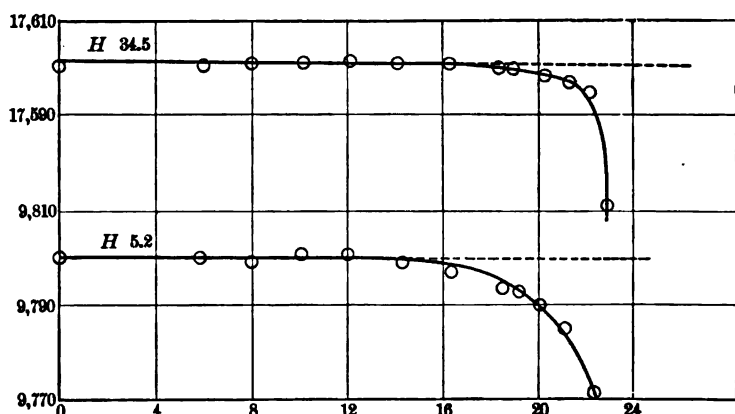


FIG. 7.—MATERIAL 1—IRON, ANNEALED. ABSCISSAS—STRESSES IN KILOGRAMS PER SQUARE MILLIMETER. ORDINATES—INDUCTIONS IN GAUSSES.

in the induction begins in both cases (for both values of the field strength) at values of the stress lower than the yield point, and that, in the case of the induction for the lower magnetizing force, this change begins at about 14 kg. per square millimeter, the elastic limit. It is probable that differential measurements of the magnetic induction in this way would afford a more sensitive criterion of a plastic deformation in iron or steel than measurements, in the usual way, of the length or elongation.

It is well known that after iron has been stressed beyond the yield point, and thereby elongated, it does not regain, for some time, its elasticity, the elastic limit becoming practically zero. The iron in this temporary state will show hysteresis in its stress-strain curve; the loading curve and the unloading curve do not coincide. Measurements were made to determine whether the magnetic induction was, under these circumstances, subject to the same hysteresis. For this purpose a specimen was loaded and unloaded cyclically, and at each step the

inductions B were measured for several values of the field strength. It was found that as long as the material under test possesses an elastic limit, and the upper value of the stress applied was not greater than this limit, no difference or hysteresis in any part of the induction cycle for a constant field strength could be detected; any such difference was not in any case greater than from 5 to 7 gaussses, a matter of 0.07 per cent., which was of the same order of magnitude as the error of measurement in

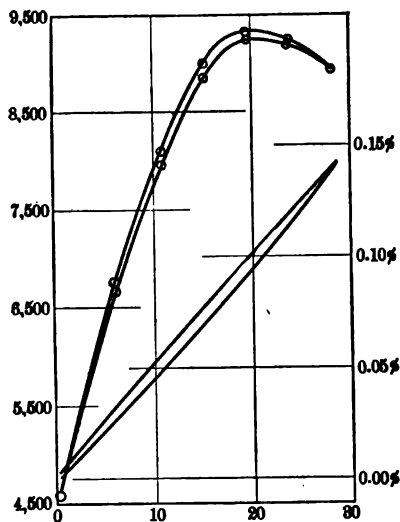


FIG. 8.—MATERIAL 1—IRON. ABSCISSAS—STRESSES IN KILOGRAMS PER SQUARE MILLIMETER. ORDINATES—INDUCTIONS IN GAUSSSES; ELONGATIONS IN PER CENT.

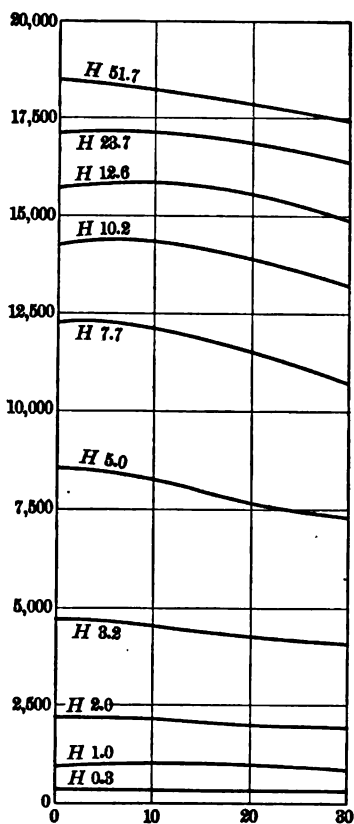


FIG. 9.—MATERIAL 1—IRON AS DELIVERED, HOT-ROLLED. ABSCISSAS—STRESSES IN KILOGRAMS PER SQUARE MILLIMETER. ORDINATES—INDUCTIONS IN GAUSSSES.

this case. The induction cycle, as well as the cycle of elongations, was perfectly reversible. When, however, the iron was stressed over the yield point, and then immediately tested, both the elongation and the induction curves showed hysteresis. The results of one such test are shown in Fig. 8, in which the abscissas represent the stresses and the ordinates, in one case, the elongations, and in the other, the magnetic

inductions B for loading (the "up" curve) and unloading (the "down" curve). These curves show that the maximum lag in the elongation is about 0.005 per cent., and in the induction, about 1.0 per cent. This bar was tested again, in the same way, 24 hr. after being overstrained; the hysteresis of both kinds had completely disappeared.

The results presented and discussed in the last two paragraphs show that, at least for the material tested, the magnetic induction (and permeability) is not dependent upon the previous elastic history of the

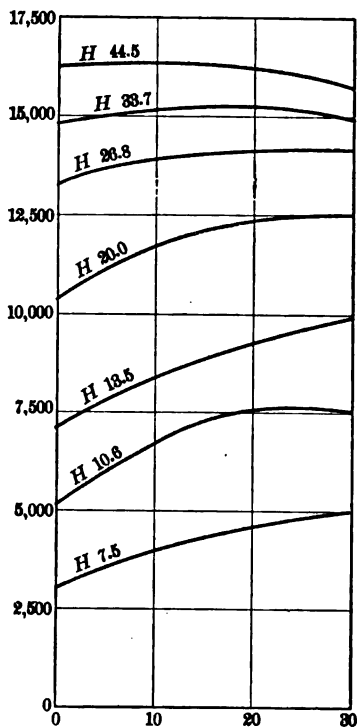


FIG. 10.

FIG. 10.—MATERIAL 1—IRON, QUENCHED FROM 950°C. ABSISSAS—STRESSES IN KILOGRAMS PER SQUARE MILLIMETER. ORDINATES—INDUCTIONS IN GAUSSSES.

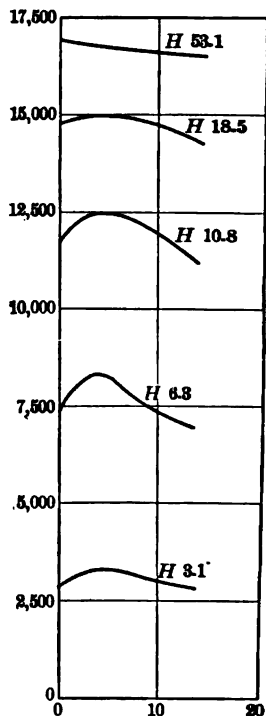


FIG. 11.

FIG. 11.—MATERIAL 2—WROUGHT IRON. ABSISSAS—STRESSES IN KILOGRAMS PER SQUARE MILLIMETER. ORDINATES—INDUCTIONS IN GAUSSSES.

material, as long as (1) the variations or alternations of stress have been within the elastic limit, and (2) the number of such alternations has been low. The present study did not include the study of the effect of thousands or millions of stress cycles upon the magnetic constants, as was originally intended. The magnetic method is, therefore, apparently not suited, as it was hoped that it would be, for the determination of the mechanical history of a steel or iron specimen, so long as the stresses in such specimens have been at all times within the elastic limit. If the elastic limit has

been exceeded, and in particular if the yield point has been exceeded, the possibility is given of subsequently determining this fact by the aid of magnetic measurements, such as have been here described.

It was suspected that, since iron after having been stressed beyond the yield point is in a metastable state, indication of this fact would be given by measurements of the magnetic permeability. Specimens were therefore so stressed, the stress removed, and the permeability, or induction, measured for certain values of the field strength, after various intervals of time. One bar showed, for certain medium values of the field strength, a decrease of induction of from 40 to 70 gaussers after 21

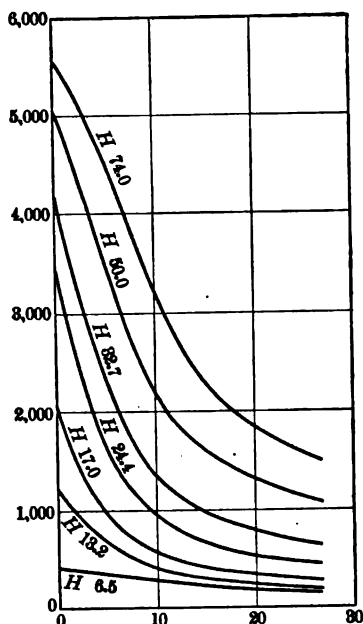


FIG. 12.—MATERIAL 3—NICKEL. ABSCISSAS—STRESSES IN KILOGRAMS PER SQUARE MILLIMETER. ORDINATES—INDUCTIONS IN GAUSSES.

hr.; and the same total change after 24 days. The magnetic change in an overstrained iron specimen takes place, therefore, in that interval of time in which the iron becomes again elastic, i.e., in the period of elastic recovery.

In Figs. 9 and 10 are shown curves illustrating the effect of stress upon the inductions B in the material 1, as delivered (hot rolled), and as annealed and quenched in water, respectively. It can be seen that, in the case of the material as delivered, there is great irregularity in the induction curves. Often the maxima occur at higher values of the stress for high than for low values of H . In both cases, the material is in a state of internal stress due to the manufacture and heat treatment, and it is

to the presence of these stresses that the irregularities in form of the curves are to be ascribed. Strangely enough, the maxima in the induction curves in the latter case are displaced toward higher stresses than in the case of the same material annealed, although the elastic limit is much lower. The elastic limit of the material 1, as delivered, was about 28 kg. per square millimeter. Owing to the presence of these internal initial stresses, it is impossible to trace any such relations between the magnetic

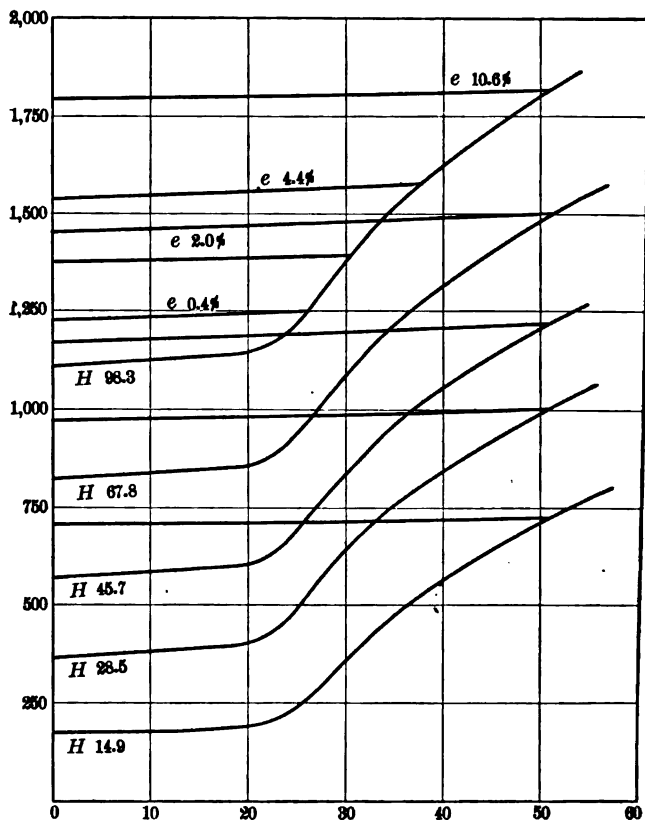


FIG. 13.—MATERIAL 4—NICKEL STEEL. ABSCISSAS—STRESSES IN KILOGRAMS PER SQUARE MILLIMETER. ORDINATES—INDUCTIONS IN GAUSSES.

and the mechanical quantities as were noticed in the case of the annealed material.

In Figs. 11, 12, and 13, are shown similar curves for materials, 2, 3, and 4, respectively, as annealed. The induction maxima, in the case of the wrought iron, occur at lower stress values than in the case of material 1, and the elastic limit lies also lower, at about 11 kg. per square millimeter. This again bears out the contention that there is a more or less simple relation between these quantities.

The two latter cases, in which no maxima occur, are interesting. The elastic limit and yield point fell together, at 23 kg. per square millimeter, for the nickel steel; the elastic limit for the nickel specimen, at 24.8, kg. per square millimeter, and the yield point at 30 kg. per square millimeter.

The position of the yield point and elastic limit is shown nicely in the induction curves of the nickel steel by the bend in the curve. The induction curve, for stresses within the elastic limit for this material, annealed, and either before or after elongation, is linear, with a slight slope upward. When the elastic limit is reached, there is a marked increase in the induction, which remains after removal of the stress.

RÉSUMÉ

1. Simultaneous magnetic and mechanical measurements were made in the tensile-testing machine on a soft iron and other materials. These were made with a view to determining whether relations existed between the physical and the magnetic constants of such materials.

2. It was found that the permeability of iron is independent of the previous elastic history of the material if (1) the alternations or cycles of stress have been within the elastic limit, and (2) the number of alternations has been small.

3. It was found that there is a close and nearly linear relation between the elastic limit of soft iron and the values of the stress at which the permeability maxima occur, as the material is elongated.

4. The magnetic indications and accompanying features of mechanical overstrain were studied.

5. The conclusion is reached that the magnetic method is applicable to the study of the previous mechanical history of such annealed iron as was investigated, since indications are thus given of the values of the maximum stress to which it has been subjected, provided that this stress has been above the elastic limit. The permeability of this material is not affected by stress applications under the elastic limit.

6. The effect of initial stresses upon the form of the induction-stress curves was noticed in the case of material as received, before annealing, and as heat treated by quenching in water.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Washed Metal

BY HENRY D. HIBBARD,* B. S., PLAINFIELD, N. J.

(New York Meeting, February, 1916)

THE process of making washed metal now followed is in principle that described by Holley before this Institute in 1879, as the Krupp washing process. Sir Lowthian Bell also experimented extensively in England during 1877 and 1878 on refining iron and joined his interest with Krupp, the process being often referred to as the Bell-Krupp process. The only plant in this country now operating this process is that of the Brier Hill Steel Co., Youngstown, Ohio. This process may be considered as the sole survivor of the numerous methods conceived for refining crude iron, when the removal of phosphorus from steel-making material was the great desideratum affecting large districts. The east coast of England and all northern Germany, the homes of Bell and Krupp, respectively, could at that time make steel only from imported ores. These processes were in use or in various stages of development 40 years ago and all except the Bell-Krupp received their quietus from the development of the basic steel processes, both pneumatic and open hearth. The aim of the early refining processes was, first, the removal of the silicon, which could be done on a sand bottom, and later, the removal of the phosphorus, requiring a bottom which would give or permit a basic slag. The incidental advantages of increased yield and small fuel consumption were not generally recognized or considered; the aim of the later operations was the saving of the difference in value between low and high phosphorus irons, as well as hastening the removal of the metalloids.

The chemistry of the washing process follows in part that of the puddling process, and indeed washing might be considered as the first half of puddling, since all the non-ferrous elements of the crude iron, except the carbon, are eliminated and the iron is maintained molten until cast. In puddling, the iron loses its carbon as well, and because of that, its fusion point rises above the hearth temperature; it freezes into a pasty malleable form, and is worked at once by squeezer, hammer, or rolls.

Since Holley's paper, important improvements have been made from time to time in both plants and methods for washing iron, and the

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degree of purification attained is far greater than Krupp accomplished particularly as to phosphorus, though Bell's results approximated those of the present practice. The costs have been diminished as well.

The use of washed metal is limited. It is in request now only as an exceedingly pure iron and is used, first, in acid steel processes, solely for the reason of its purity, as in open-hearth and Bessemer steels of high grade; and, second, because of its high carbon contents (3.25 to 3.50 per cent.) as well as purity, as a carburizer in crucible steel.

Washing Plant of Brier Hill Steel Co.

The washing plant of the Brier Hill Steel Co. (Fig. 1) is placed near one of its blast furnaces, which supplies it with molten crude iron. It consists briefly of a reservoir, a Pernot furnace, a reheating furnace, and a casting machine.

The Reservoir.—The reservoir is about 100 ft. or more from the blast furnace, a runner in the floor of the cast house conducting the stream of molten crude iron to it from the furnace. It has a brick lining 18 in. thick, is of 50 tons capacity, and is mounted on top of the ram of a vertical hydraulic cylinder, which straddles a 100-ton weighing scale. It is provided with mechanism for tilting to pour the metal. Fifty tons of iron make four washed-metal charges, which take about 1 hr. each to work, and, since the blast furnace is tapped every 4 hr., the reservoir provides a continuous supply of iron to the washing plant. The reservoir receives iron when at its lowest position, is raised by the hydraulic plunger about 15 ft., and discharges to a short open runner which conducts the molten crude iron to the Pernot furnace. Iron has been held molten in the reservoir 48 hr. with the help of a small oil jet, and then washed without difficulty.

The Washing Furnace.—The Pernot furnace in which the washing is done (Fig. 2) differs in many details from the Pernot furnaces which have been described. In fact, nothing of the original furnace remains and hardly any of the details of the original design, though the general plan is the same. The changes made have resulted in greater durability and efficiency, but even now, the cost of repairs is an important item in spite of the relatively low temperature of the operation.

The pan, as the circular hearth is termed, was originally designed to revolve on wheels, but now runs on conical rollers, which are kept spaced by a spider. The rollers travel on a circular track below the pan, turning in the same direction as the hearth, but at half the speed. On the bottom of the pan is a circular track on which it travels over the rollers. The pan is driven by a bevel pinion engaging a gear on the bottom of the pan. The pan lining is so thin (8 in.) that water is continuously sprayed upon the bottom, from underneath, to keep it cool.

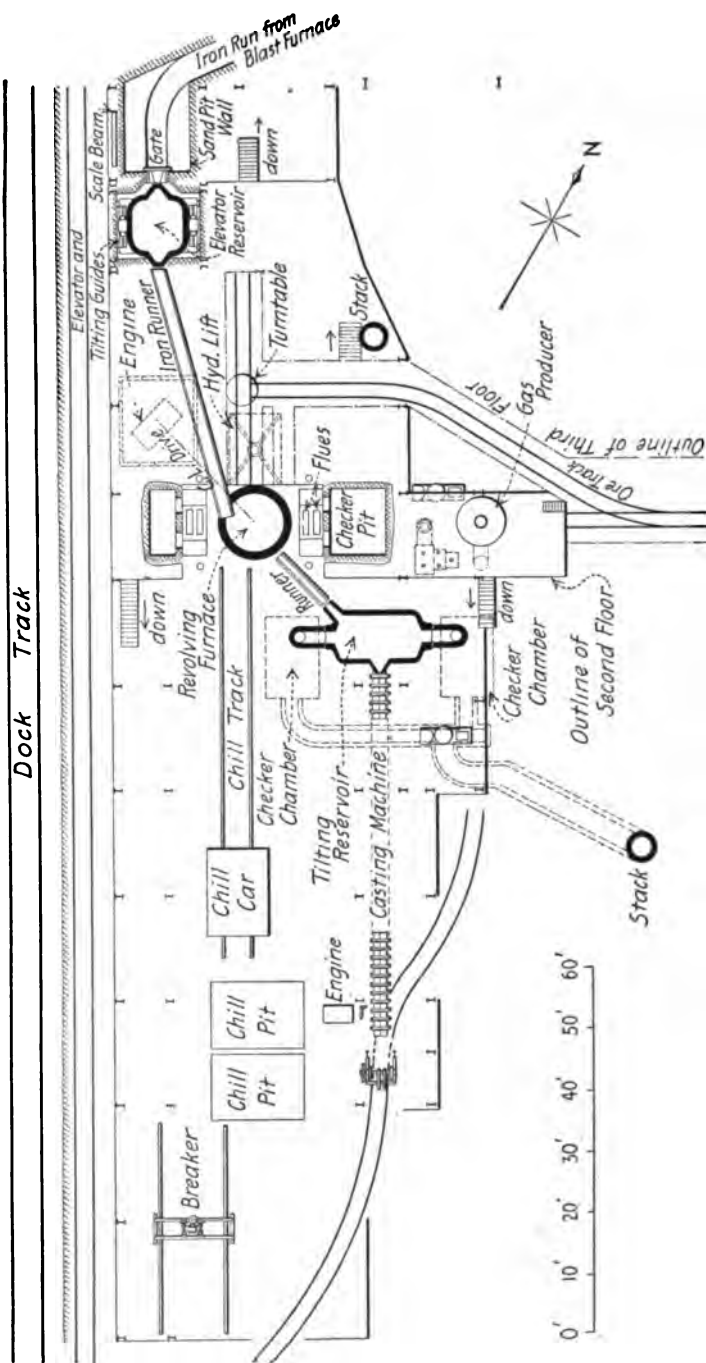


FIG. 1.—WASHED METAL PLANT OF BRIER HILL STEEL CO.

In the superstructure, at one side of the single operating door on the charging side and above the pan, is an opening with which the crude-iron

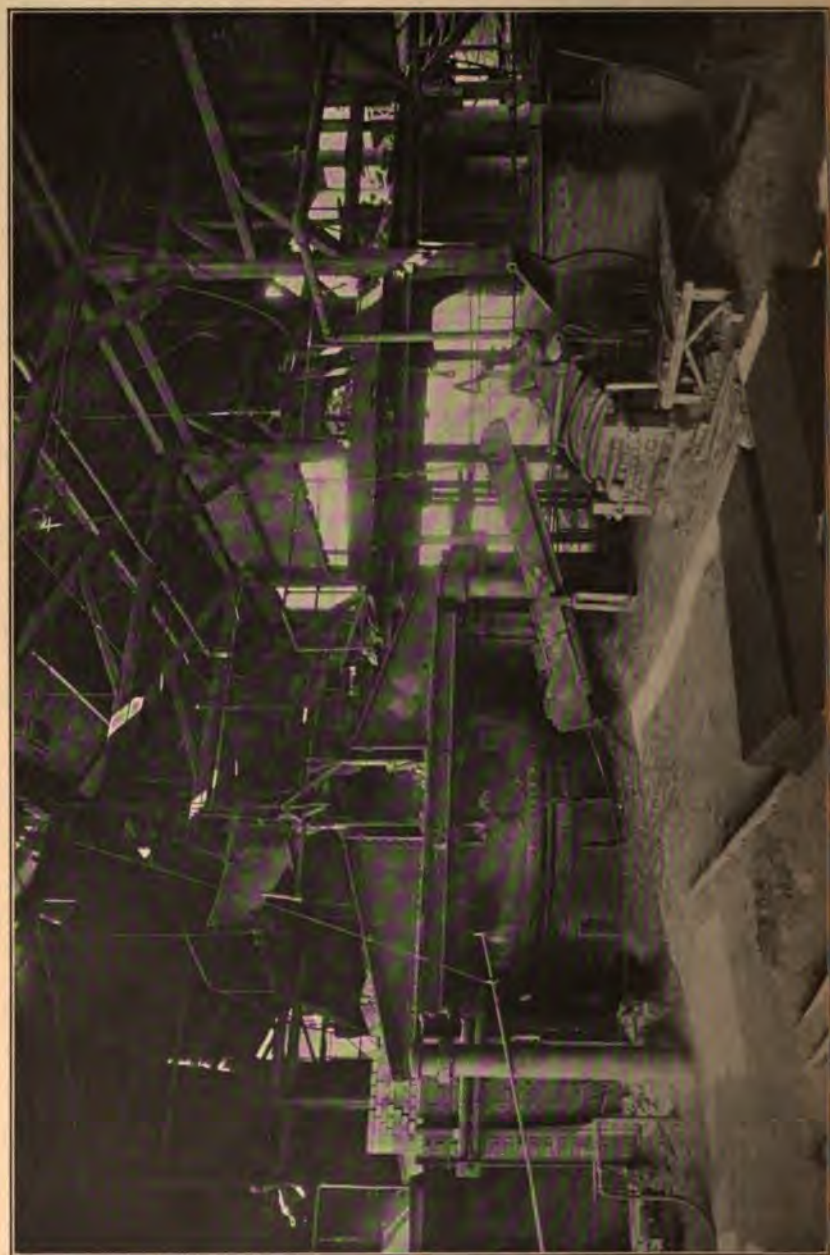


FIG. 2.—FRONT VIEW OF WASHING FURNACE, SHOWING TIPPING HOIST FOR HOLDING WASHED METAL WHILE CHARGED IS MASHED IN THE FURNACE.

runner from the reservoir is connected and through which the molten iron is charged into the furnace. In the side of the pan near the bottom

is a tap hole and near the top a slag notch. A hole through the furnace roof connects with an ore chute through which the ore is shot for fettling the hearth after each charge has been drawn off.

One circular gas producer supplies gas for the use of the furnace. The ability of a single producer to run this plant, depends on the fact that the coal used contains only about 1.5 per cent. ash, with 44 to 46 per cent. of volatile matter, and will not coke under any circumstances. This makes it possible to carry a fire 8 ft. or more deep, and drive at a high rate with a steam injector for the air, without having above 2 per cent. CO_2 in the gas. This coal in earlier days was used raw in the blast furnaces of the neighborhood without other fuel for making pig iron and is in this respect like the Scotch coals.

A section of the floor, 10 by 10 ft., on the charging side of the furnace, is arranged to be lowered, which is done after each heat is run out, providing a place for the men to stand to repair the tap hole and cinder notch, the pan being revolved so that these openings are on the high side.

Operation.—After a heat has been run out, the furnace is drained; any holes in the hearth are puddled out and filled with iron ore, and the fettling, consisting of dampened ore, is run down in successive batches through the roof, the hearth being revolved a little at each dose, so that the bank of fettling is continuous around the hearth. The gate of the ore chute above is controlled by the melter from the working platform by means of overhead levers. While the banks are thus being built up, the bottom is covered with ore thrown by the shovelful through the operating door. The ore used for bottom making and fettling is a high-grade specular hematite, broken into pieces from the size of a hen's egg down, with enough fine ore to make a compact mass without voids.

Between 20 and 30 min. are consumed in repairing the bottom and fettling. The tap hole and cinder notch are cleaned out and plugged with sand.

The firing is then resumed to heat and set the fix, the reservoir is raised, and 12 tons of iron is run into the pan, which is then started revolving. A reaction immediately takes place between the oxygen of the ore and the silicon, phosphorus, and manganese of the iron, the bulk of these elements being eliminated in a few minutes, by passing into the slag which is formed. The small percentage of carbon which goes off during the operation (3 to 5 per cent.) makes enough CO gas to give a vigorous boiling action, throwing jets of slag 3 to 5 in. above the bath at first when the temperature of the bath is greatest.

About 15 min. after the crude iron has been charged the slag is made to run off through the slag notch. This slag contains most of the oxidized silicon, phosphorus, and manganese from the crude iron, and perhaps a quarter of the sulphur. A second slag is then formed by additions of lime, which, with the ore melted from the fettling, takes up more of the

phosphorus and sulphur of the iron. These slags are rich in iron and are therefore smelted afterward in the blast furnaces for making basic pig.

Tests are taken from time to time, averaging about five to a heat, to determine the degree of dephosphorization. The test is a circular cake about 5 in. in diameter and 1.25 in. thick in the center, being thinner at the edges. This cake is broken into pieces. The test contains a myriad of small bubbles and has a faint crystalline or columnar structure normal to the bottom surface. The lower the phosphorus, the larger the bubbles, and the fainter the crystalline structure mentioned. With very low phosphorus, bubbles as large as $\frac{1}{16}$ in. in diameter may be found within $\frac{1}{4}$ in. of the lower surface, with smaller ones still lower and larger ones up to $\frac{1}{4}$ in. in diameter near the upper surface.

The whole washing operation takes about an hour, the most of this time being consumed in securing the extremely low phosphorus content required. When ready for casting, the tap hole is turned to the proper position on the pit side of the furnace and is then dug open by means of a hand bar.

The metal runs from the furnace fairly well, but looks cool, almost of an orange temperature, perhaps not over $1,250^{\circ}$ C. It gives off copious brown fumes, as well as scintillating sparks and CO gas, which burns, the whole making a brilliant effect and, when viewed from a distance, gives one the impression that the metal is hotter than it really is.

Casting.—For casting the metal two means are provided: The one generally used when the product is for domestic consumption is a broad shallow cast-iron pan on a car, which runs on a track leading from the Pernot furnace down the center of the cast house. The charge is run out into this pan, where it forms a single plate from 4 to 6 in. thick, and the car is then drawn away from the furnace to cool. Water from a hose is played upon this plate for a few minutes, while hot, to loosen any slag which may come out after the metal and is not trapped off while casting. When the cake of metal is cool enough to be moved without breaking, but is still red hot, it is lifted from the car with a crane and put in a pit. Water is then turned on and it is flooded until cold enough to handle, when it is broken up into one-man size chunks, either with a sledge or by a traveling breaker. This is not difficult, because the water cooling has filled it with cracks, which go clear through the plate. These cracks seem to have no system, except that they are always at right angles to the upper and lower surfaces of the plates, leaving the pieces very irregular in outline.

The other casting method, usually employed for export orders, is to run the washed metal without the slag, into a 12-ton tilting, oil-fired regenerative furnace, located at a lower level, where it is held about 15 min., heated, and poured directly into the molds of an endless-chain casting machine, which moves at the rate of about 15 ft. per minute.

The joints between the tilting hearth and the stationary ends of the reheating furnace are packed with asbestos, which closes them tightly and will endure the moderate temperature, say 1,350° C., fairly well.

The metal is a little hotter than when it left the Pernot furnace, but is still of an orange heat. In the molds the metal is puffed up by gases being evolved, but settles more solidly soon after being cast, leaving a fin standing up all around each pig. For casting into this form, an extra charge is made.

Yield.—The yield of washed metal is about the same as the weight of the original crude iron charged.

Chemistry of the Washing Process

The crude iron contains: Silicon, 1 to 1.25 per cent.; sulphur, 0.02 to 0.03 per cent., and phosphorus, 0.090 to 0.10 per cent. All of the silicon and manganese, 90 to 95 per cent. of the phosphorus, and 30 per cent. of the sulphur are eliminated. The silicon and manganese with 75 to 80 per cent. of the phosphorus leave the iron in 10 min. and 30 per cent. of the sulphur in 15 min. The silicon, manganese, and phosphorus are removed by oxidation, while the sulphur seems to be eliminated by liquation, the sulphide of iron getting into the slag mechanically without oxidation. At certain high temperatures, the affinity of sulphur for oxygen seems to be weaker than its affinity for iron. The slag takes sulphide of iron from the metal until equilibrium is established between the proportions of sulphide in the two, when no more will pass. When a second slag is formed, some of the sulphide will enter that, making the elimination from the metal more complete. Should the slag have too much sulphide, as has happened when some blast-furnace slag got in with the crude iron, some of the sulphide will pass from the slag into the metal, and in that way establish the equilibrium.

Individual heats of washed metal have been made having as little as 0.002 and even 0.001 per cent. of phosphorus, and it has been furnished on specification of 0.006 phosphorus and under.

When iron heavily charged with carbon, like washed metal, lies molten in contact with molten oxide of iron, there will be, if the heat is high enough, a continuous oxidation of carbon to CO until the temperature falls to a certain degree, when the action will cease. That reaction is endothermic and the consumption of heat automatically brings the temperature down to that point. This temperature has not been determined, but there is ground for suspecting it to be near 1,300° C.

Analyses of Metals and Slags

Typical analyses of crude irons with washed metals and slags therefrom are given below.

	Pig Iron, Per Cent.	Washed Metal, Per Cent.
Silicon.....	1.25	none
Phosphorus.....	0.095	0.010
Sulphur.....	0.020	0.015
Manganese.....		none
Combined carbon.....		3.40

Final Slags

	Per Cent.
SiO ₂	16.50
MnO.....	6.50
CaO.....	8.50
P ₂ O ₅	1.05
S.....	0.075
FeO.....	50.25
Fe ₂ O ₃	16.90
	99.775

THE REMOVAL OF METALLOIDS IN THE PIG-WASHING PROCESS

BY EDWARD L. FORD, YOUNGSTOWN, OHIO.

In the accompanying diagram (Fig. 3) showing the removal per cent. of metalloids in washing, the rate of elimination of each seems to be constant for about the first 10 min., and (with the possible exception of sulphur) the angle which the line of each makes with the base line probably represents the relative net heat results of each complete reaction. I use the word "complete" because the net result is the sum of a number of things: In the case of silicon, for example, the heat + or - from Si to SiO₂; from FeO to Fe; from Fe₂O₃ to FeO; the formation of silicate by the silica; the liquefying of this silicate, etc.

Why these rates should not remain constant up to complete elimination is due to a number of things—changes in temperature, changes in character of atmosphere, changes in composition of slag and resultant metal, and very possibly other causes. We have worked out some of these; and, although they may not be the only factors, they seem to explain fairly well what happens.

Silicon.—The rate of silicon removal is constant to about 95 per cent., when it begins to become less rapid. This, I think, is due entirely to the want of perfect mobility of the slag and metal, and therefore to the fact that the silicon is not present in equal amount in all parts of the metal at this stage of the process. In fact, it must take an appreciable amount of time for the metal in the center of the mass to reach iron oxides on its surface.

Manganese.—Manganese seems to behave in the same way, although the rate of removal is a little less rapid. Its retardation at the end is due to the same causes.

Phosphorus.—The phosphorus removal-rate is constant to about 70 to 75 per cent., when retardation sets in, and progressively increases, the line tending to parallel rather than intersect the 100 per cent. line. The long time required to remove phosphorus, as compared with silicon and manganese, would make the lack of mobility at this stage a factor of practically no consequence; but, after the elimination of silicon, carbonic oxide begins to be formed; and (as Pourcel showed very clearly, in the early days of the basic Bessemer process, in the discussion on the necessity for the overblow to remove the phosphorus) phosphorus cannot be removed from iron when CO is being formed intermolecularly in quantity; and no doubt the proportion of phosphorus removed is inversely related to the amount of CO formed, during a given period. True, the forma-

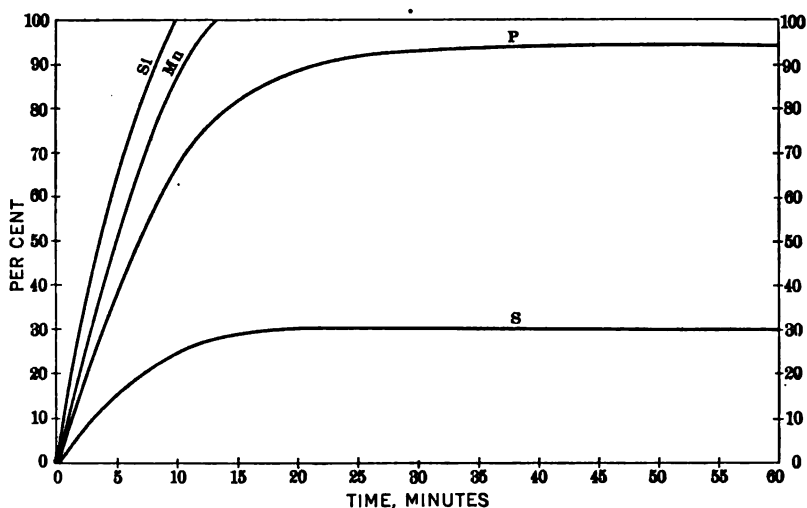


FIG. 3.—REMOVAL OF METALLOIDS IN THE WASHING PROCESS.

tion of CO, by lowering the temperature of the bath, tends to stop itself; but it completely ceases only when the metal is so cool that it begins to set; and this, of course, impairs the fluidity necessary to bring new surfaces of metal in contact with the slag, in order to oxidize the small amount of phosphorus left. We, therefore, finally reach a point of equilibrium when the CO formed just reduces the amount of phosphorus oxidized. If we increase the heat of the furnace, to increase the fluidity of the metal, we increase again the amount of CO that can form, and so are no better off.

Sulphur.—Our theory of the elimination of sulphur I have never seen formulated and would hesitate to report it here, if I were not trying to give the *rationale* of the washing process, as we have known it. Even at quite a low temperature, sulphur appears to form sulphides, preferably

to oxygen compounds; and as the temperature increases, this tendency seems more marked. In neither the blast furnace, the converter, the open hearth, or the puddling furnace, nor in our washing process, is the sulphur removed, except as a sulphide. In roasting sulphurous iron ores, however, it does go off as SO_2 ; but this process, as is well known, is performed at as low a temperature as possible. Our view is that for any given temperature, atmosphere, slag, and metal composition, the amount of sulphide in the slag bears a constant ratio to the amount in the metal. That there is an equilibrium, so to speak, which cannot be altered without a change in one, at least, of the controlling factors above mentioned. In our own process, as carried out by us, the ratio seems to be one in the metal to five in the slag, or, to put it another way, a slag with 0.10 per cent. sulphur will give a metal with 0.02, and to get 0.01 per cent. sulphur in the metal, the slag must not have more than 0.05 per cent. As corroborating this proposition, the two following incidents may be interesting:

On one occasion, pig iron with 0.025 per cent. sulphur was run into the washing furnace at a time when an accident at the blast furnace had let some blast-furnace cinder get into the receiver together with the pig metal. Some of this cinder ran with the metal into the washing furnace. The resulting washed metal carried 0.04 per cent. sulphur. With our ordinary practice this would have been about 0.017 per cent.

Some washed metal with 0.015 per cent. sulphur was melted in a cupola with coke. The resultant metal ran 0.10 per cent. sulphur. Some white iron with 0.10 per cent. sulphur was melted in the same cupola. Result: sulphur, 0.11 per cent. We had on our yard a heat of washed metal, containing 0.09 per cent. sulphur. This was melted in the same cupola. Result: sulphur, 0.12 per cent. These results seemed to point to from 0.10 to 0.12 per cent. as the equilibrium point in the resultant metal from these operations, no matter what sulphur the metal contained to start with.

With the above statement of our sulphur theory, the following is our explanation of the sulphur line in the diagram:

This sulphur line probably has only this relation to those of the other metalloids, that it tends to become tangent always directly below the point of complete silicon elimination, because at this time ends the formation of new sulphur-free slag, capable of taking up more sulphur from the metal. The percentage removal, therefore, varies with the sulphur contents of the iron, in contradistinction to the other metalloids, the percentage removal of which is constant.

Low-sulphur washed metal is the only marketable product; hence we aim to keep the sulphur in our pig iron as low as we can, with low-silicon iron. Our "fix" ores contain a little sulphur; the gas from our producer carries a little; and our limestone a little; so that the resulting slag, if none

came from the pig iron, would run about 0.03 to 0.04 per cent. as a minimum. Our practice shows us that, in a metal containing 0.007 per cent. sulphur, this would give us no removal.

If we could obtain raw slag materials absolutely free from sulphur, our washed metal would be 0.007 per cent. lower in sulphur with our regular pig iron than it is now. If we were satisfied with 0.05 per cent. sulphur in our washed metal, we could use a pig so high in sulphur that the amount taken out would probably approximate 60 per cent., rather than 30, as now; for when we formed our second slag, it would be capable of taking up a considerable portion of this remaining sulphur, before reaching the equilibrium point.

If the other conditions incident to the process permitted it, we could, by making new slags, reduce any amount of sulphur in the pig to about 0.01 per cent., but we could never get much lower than that, no matter how low the pig was, as long as our slag-making materials were as high in sulphur as they are now.

Our experience teaches us, however, that the blast furnace is the best and cheapest desulphurizer.

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Manganese-Steel Castings in the Mining Industry

BY WALTER S. MCKEE, * CHICAGO, ILL.

(New York Meeting, February, 1916)

WHEN mixed with common cast steel in quantities ranging from 11 to 13½ per cent. and properly treated, manganese increases the ductility of the metal and adds greatly to its toughness and resistance to abrasive action, which properties comprise the distinctive characteristics that make manganese steel so valuable for many purposes at the present time.

Manganese steel has been manufactured for about 26 years. It was first made in a small way in England and subsequently produced in the United States in 1892. At present the manganese-steel foundries of this country have an annual capacity of about 60,000 tons and most of the growth has occurred within the past 10 years. In the early days of the industry it was impracticable to make manganese castings weighing over a few thousand pounds, but at present crusher heads, rolling-mill pinions, and similar castings weighing up to 30,000 lb. each are produced, and it does not appear that the limit of weight has been reached by a considerable margin.

Manganese steel is similar in analysis to ordinary first-class converter metal except that it is high in carbon and contains from 11 to 13½ per cent. of manganese. From one standpoint manganese steel is similar to malleable iron in that the casting as it comes out of the sand is hard and brittle and must be made ductile by a heat-treating process. The heat treatment is an essential part of the process of making manganese-steel castings and must be properly carried out to secure satisfactory results. The necessity for heat treatment limits the thickness of sections which may be made of manganese steel. Originally the annealing depth was comparatively small, but researches carried on during the past few years have resulted in an increase in the thickness of section up to 5½ in., which it is possible to treat successfully. Large castings are cored out to a minimum thickness of 5½ in., thus not only eliminating some of the internal stresses which are invariably set up in a very thick casting, but also reducing the weight and inertia of the piece, an

* Vice President, American Manganese Steel Co.

important consideration if the casting is to be a moving part in a machine. At the same time, the thickness of $5\frac{1}{2}$ in. is sufficient to permit the use of manganese steel for the heaviest classes of machinery.

Before taking up the application of manganese-steel castings to machinery used in the mining industry, a few details covering the chemical and physical characteristics as well as the micro-structure of this material will be presented. Chemically, cast manganese steel has about the following composition:

	Per Cent.
Carbon.....	1.25
Silicon.....	0.30
Manganese.....	12.50
Sulphur, less than.....	0.02
Phosphorus, about.....	0.08

The following results are an average of 19 physical tests of manganese steel: Elastic limit, 53,396 lb. per square inch; tensile strength, 108,460 lb. per square inch; elongation in 2 in., 33.71 per cent.; reduction of area, 38.56 per cent.

The distinctive characteristics of manganese steel are clearly indicated by the microscope. As is well known, manganese is present in all ordinary steel, but the metal does not become austenitic until about 6 per cent. manganese and 0.80 per cent. carbon are introduced. Commercial austenitic manganese steel generally contains from 11 to 13.5 per cent. manganese and from 1.00 to 1.30 per cent. carbon.

In the cast state, manganese steel is composed principally of austenite and free cementite, austenite being a solution of iron, manganese, and the carbide of iron and manganese, while free cementite is composed of the carbides of iron and manganese which remain undissolved. Free cementite is hard and brittle, therefore manganese steel is also brittle in the cast condition. We thus have a metallographic explanation of the chief characteristics of manganese steel. Microscopic examination reveals the fact that if the steel is heated to a proper temperature and quenched in water, the free cementite is dissolved and the whole becomes austenite.

The accompanying micro-photographs, Figs. 1 to 5, show the structures of manganese steel as cast and after treatment. Fig. 1 shows the structure of manganese steel in the cast condition. Fig. 2 shows the structure of the same steel heated to a temperature of 1,750°F. and quenched in water. The heat treatment was not correct, and although much of the free cementite has disappeared, some of it still remains in chunks as shown by the dark spots. Fig. 3 shows the structure of the same steel heated to a temperature of 1,800°F. and quenched in water. In this case the heat treatment is correct and all of the free cementite has disappeared, leaving a purely austenitic structure.

The effect of subsequent heating and cooling in air is shown in Figs. 4 and 5. Fig. 4 shows the structure of a manganese steel which had received a proper heat treatment and was afterward heated to a

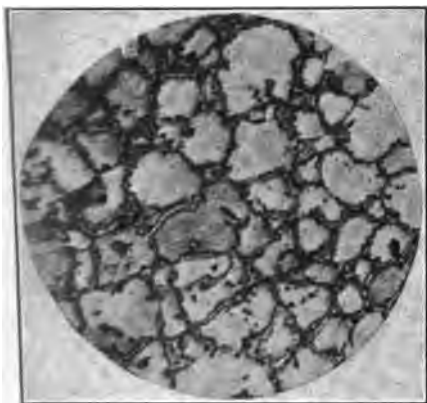


FIG. 1.—STRUCTURE OF MANGANESE STEEL IN CAST CONDITION. $\times 80$ DIAMETERS.

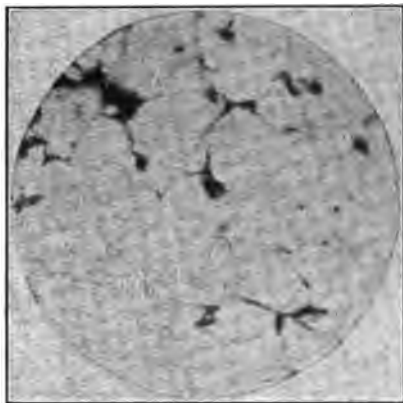


FIG. 2.—THE SAME STEEL INCORRECTLY HEAT TREATED, SOME CEMENTITE VISIBLE.

temperature of 700°F . and allowed to cool in air. The structure is exactly similar to that shown in Fig. 3; the steel, therefore, has not been injured. Fig. 5 shows the structure of the same steel heated to



FIG. 3.—THE SAME STEEL PROPERLY HEAT TREATED, RESULTING IN A PURE AUSTENITIC STRUCTURE. $\times 80$ DIAMETERS.

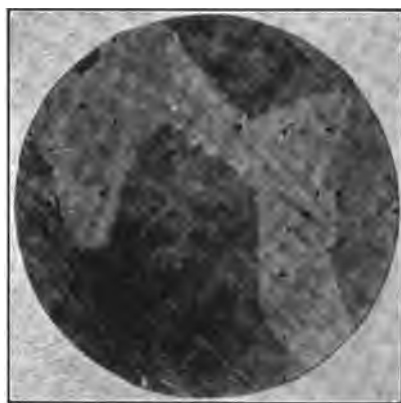


FIG. 4.—MANGANESE STEEL CORRECTLY HEAT TREATED AND SUBSEQUENTLY HEATED TO A TEMPERATURE OF 700°F . AND COOLED IN AIR WITHOUT CHANGING THE AUSTENITIC STRUCTURE. $\times 80$ DIAMETERS.

a temperature of 750°F . and allowed to cool in air. A heavy separation of cementite, shown by the needle-like particles, is clearly visible. This proves that heating the steel to over 700°F . destroys the purely austenitic

structure, seriously injuring the quality of the metal. It is therefore imperative that manganese steel should not be heated to a temperature exceeding 700°F. after being heat treated. The experiments on which the foregoing micro-photographs are based were carried out with the aid of an electric furnace equipped with a pyrometer.

By the Brinell test, manganese steel toughened or untoughened shows the moderate average hardness number of about 200. The extreme outer surface of the treated steel shows a slightly lower number than at a depth of about $\frac{1}{8}$ in., from which point the number remains constant to the core of the casting. The lowering of the hardness number at the surface is due to the oxidization of the carbon during the heat treatment.

The qualities of manganese steel have led to its adoption extensively for the vital parts of heavy-duty machinery, particularly parts subjected

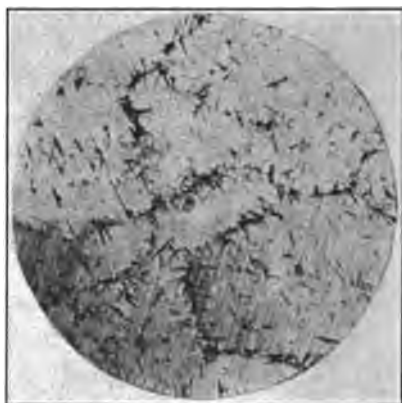


FIG. 5.—THE SAME STEEL HEATED TO 750°F. AND ALLOWED TO COOL IN AIR RESULTING IN THE PRECIPITATION OF CEMENTITE INJURING THE QUALITY OF THE METAL. $\times 80$ DIAMETERS.

to unusual wear, shock, etc. Manganese-steel castings are used today in the construction of wearing parts for the following kinds of machinery: Rock and ore crushers, coal breakers, ball mills, tube mills, pulverizers, comminutors, clay mills, traveling cranes, gold dredges, harbor dredges, steam shovels, ditching machines, pig-casting machines, pug mills, coke machines, steel-rolling machinery, stamp mills, cable haulage systems, screening apparatus, centrifugal sand pumps, grab buckets, concrete mixers, etc. Manganese steel is also extensively used in the manufacture of steam and electric railway track work, elevator buckets, gears, pinions, mine and skip car wheels, and for numerous other purposes.

The use of manganese steel for tube-mill liners is a recent development which is of special interest at this time because of the curtailment in the supply of lining materials previously used, by reason of the war abroad. Until the introduction of manganese-steel liners, tube mills

were lined almost exclusively with silex blocks imported from Europe, principally from Denmark. The pebbles used in tube mills also have been imported, chiefly from France and Denmark. The manganese-steel liners have been developed to the point where they form a satisfactory substitute for silex blocks, and experiments are now being conducted on the use of 2- and 3-in. diameter manganese-steel balls in place of imported pebbles. A manganese-steel tube-mill liner known as the Komata, developed by E. C. Brown, is now used with good results.

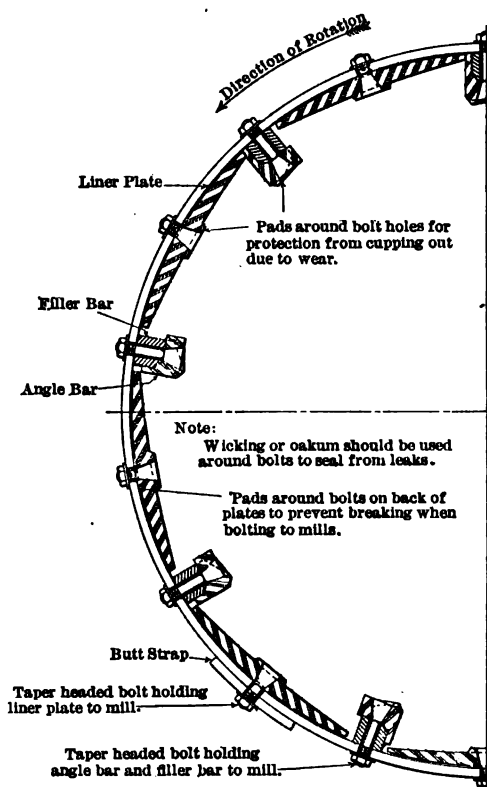


FIG. 6.—HALF-SECTION OF A TUBE MILL FITTED WITH MANGANESE-STEEL KOMATA LINING PLATES AND LIFTING BARS.

As indicated in Fig. 6, it consists simply of a series of plates and lifting bars securely attached to the shell of the mill by square-headed taper bolts, drawn up from the outside, so that the plates are held in place even when worn extremely thin. The lifting bars perform an important function, their action being similar to the bars installed in Bradford coal breakers. The bars carry the pebbles up to a point considerably beyond the center line of the mill, causing an almost perfect cascading action. For this reason the pebbles maintain a spherical shape and the great com-

motion and rolling action of the pebbles produces an effective grinding action. The pebbles do not slip on the liners, with the result that the life of the plates is prolonged and the tendency of pebbles to become flat is prevented.

The lifting bars are invariably made of manganese steel because of the great wear to which they are subjected. The lining plates are preferably made of the same material, but if local circumstances warrant, they can also be cast of less expensive semi-steel or white iron at local foundries. To prevent the liner from cupping around the bolt holes, raised bosses are placed on the castings around each hole.

In operation the pulp soon gets behind the plates and forms a cushion or backing. The ribs, or lifting bars, are spaced from 18 to 20 in. apart according to the diameter of the mill. The filler bars which are placed underneath the lifting bars are not exposed to wear and therefore are made of soft cast iron. The joints in the bars are staggered with the joints of the liner plates to prevent the formation of grooves or channels in which the pulp might travel and wear the shell of the mill.

When a manganese-steel liner is employed, the speed of the mill has an important influence on the results obtained. The correct speed can be derived from the following formula:

$$\text{Revolutions per minute} = 183 \div \sqrt{D}$$

where D is the diameter of the mill in meters.

To secure satisfactory results, the pebble load should be about 3 to 5 in. below the center line of the mill, depending on the diameter. When grinding ordinary coarsely crushed battery pulp, the new pebbles fed into the mill should not exceed $2\frac{1}{2}$ in. or, in any case, 3 in. in diameter.

For the year ended June 30, 1914, one of the largest milling companies in Nevada reported the following consumption of lining material: Manganese-steel lifting bars, 16,633 lb.; semi-steel liner plates, 49,916 lb. During this period, 341,354 tons of ore were ground, comprising the discharge of 12 batteries equipped with 3-mesh screens, together with the discharge from eight batteries passed through Chilean mills equipped with 16-mesh screens, both products being passed through Dorr classifiers before the sands were fed to six 5 by 22 ft. tube mills. The actual sliming done by the tube mills was estimated at 100 tons per day per mill, 80 per cent. of the product passing through a 200-mesh screen. The pebble consumption for a similar period was 1,647,524 lb.

At this plant a set of manganese-steel lifting bars lasts about 14 months and a set of semi-steel liner plates, which are $1\frac{1}{16}$ in. thick at the center, tapering to $1\frac{1}{16}$ in. at the edges, lasts about 23 months.

In some of the mills manganese-steel liner plates also were used. They were in service 29 months and when removed weighed less than 30

lb. each. These plates were $1\frac{1}{8}$ in. thick at the center, tapering to $\frac{3}{8}$ in. in thickness at the edge.

The consumption of metal and pebbles per ton of material ground, according to the figures given previously, was as follows:

	Consumption per Ton of Material Ground, Pounds
Manganese-steel lifting bars.....	0.049
Semi-steel liner plates.....	0.137
Pebbles.....	4.83

In the mills in which the foregoing tests were made, it is not customary to renew the whole liner at once, but to remove worn-out pieces from time to time as necessity demands.

Some interesting tests of manganese-steel liners have also been made on a Schmidt-Davidson type mill in operation in New Zealand. This mill is 4 ft. 6 in. in diameter and is run at 23 r.p.m., which was found to be the most effective speed when using lifting bars. The liners in this case were of hard, white cast iron made at a local foundry. When new, the liners were 1 in. thick in the center, tapering to $\frac{3}{8}$ in. at the edges, the lining weighing 5,940 lb. There were 10 rows of ribs in the mill with an aggregate weight of 2,347 lb. The liners were used for $75\frac{1}{2}$ weeks and the angle bars for $60\frac{1}{2}$ weeks before they required replacing. From 187 to 232 tons of ore were passed through the mill daily, these figures including the coarse sand which was returned to the mill as many times as necessary. The stamp delivered an average of 76 tons of ore to the mill daily. The pebbles were from $1\frac{1}{2}$ to $2\frac{1}{2}$ in. in diameter.

During the last six months of the run, the fineness of grinding was increased and for this reason the quantity of ore fed to the mill from the stamps was reduced to 55 tons per day, exclusive of returns. The fineness of the ore before and after grinding, when crushing this tonnage, averaged as follows:

Screen Mesh	Battery Discharge, Per Cent.	Tube Mill Discharge Per Cent.
+10	6.2	
+20	6.7	
+40	20.2	
+60	8.3	
+80	7.0	
+100	5.6	0.6
+150	10.4	6.6
+200	2.4	4.7
-200	33.2	88.1
	100.0	100.0

The pulp passing through the tube mill varied between 1 and 1.4 parts of moisture to 1 of ore by weight. The adoption of this special

form of liner is said to have reduced the consumption of pebbles by 70 per cent., the liner cost having been cut 37 per cent.

A company at Tonopah, Nev., is operating two 5 by 18 ft. tube mills fitted with Komata linings. These mills run at 24.4 r.p.m. and they require 98.67 hp., or 46.335 hp. per mill. The mills are fed with the discharge from stamps which are equipped chiefly with 6-mesh screens, a few batteries being provided with 4-mesh screens. The feed contains 38 per cent. moisture. With a pebble consumption of approximately 4 lb. per ton of ore ground, the results are as follows:

Screen Mesh	Battery Discharge Per Cent.	Tube Mill Discharge, Per Cent.	Finished Product Per Cent.
+10	10.8		
+14	10.7		
+20	10.0		
+28	10.3		
+35	7.7	6.0	
+48	7.1	2.4	
+65	5.6	5.3	
+100		16.1	22.0
+110	6.0		
+150	6.6	28.3	12.2
+200	4.0	14.2	7.3
-200	21.2	33.1	78.3

Manganese steel has been found particularly suitable for dredge buckets in the gold-dredging districts of Montana, California and elsewhere, the life of the manganese-steel buckets being reckoned by years instead of months. The buckets usually are cast solid and the lips riveted in place; they range from 600 to 5,000 lb. in weight each, the average being about 2,200 lb. A number of buckets of this type have been in service since September, 1909, and judging from present indications their total life will be from 10 to 12 years.

Usually placer dredge buckets are removed from service because the metal underneath the back eye wears away. In some cases the buckets are worn entirely through to the bushing at this point before they are taken from the line. The metal underneath the back eye ranges from $1\frac{1}{2}$ to 3 in. in thickness, depending on the size of the bucket. Some observations were recently made of the wear on 50 manganese-steel buckets placed in service on a dredge in Montana in December, 1911. These buckets have a capacity of 9 cu. ft. each. When new, the metal underneath the back eye had an average thickness of $2\frac{1}{2}$ in. After having been in service for three years, actual running time, the buckets were again measured and the metal under the back eye was found to have been worn to an average thickness of $1\frac{3}{8}$ in. In other words, about $\frac{5}{8}$ in. of metal was worn away. This means that $1\frac{1}{4}$ in. of metal are still available for wear, assuming that the buckets are worn

out when the thickness is reduced to $\frac{5}{8}$ in. Since the wear has amounted to $\frac{5}{24}$ in. per year, the estimated total life of the buckets is nine years.

Another set of measurements was made recently on a line of 86 5-cu. ft. dredge buckets which have been in service in Colorado for 32 months, actual running time, covering the period from March, 1911, to Aug. 5, 1915. These buckets which when new had a thickness of $2\frac{1}{8}$ in. underneath the back eye showed an average wear for the period of $\frac{3}{8}$ in. Assuming that the 5-cu. ft. manganese-steel buckets would be discarded when but $\frac{5}{8}$ in. of metal is left underneath the back eye, the estimated life is $10\frac{1}{2}$ years running time. It is estimated that the average life of a carbon-steel bucket line on this particular dredge would not exceed 30 months.

Most dredge buckets now made are of the two-piece type, the bottom and hood each being cast integrally. With this design of bucket, leakage is avoided and there is no danger of loss of valuable material through the rivet holes while the bucket is carrying its load from pond to the upper tumbler. A short lip can be used on the two-piece buckets and comparatively few rivets are required.

The tumblers on gold dredges also are frequently made of cast manganese steel, some of them weighing 24,578 lb. The tumbler is simple in construction, consisting of nothing more than spool cast in one piece, the bore being finished by grinding. Renewable wearing plates are eliminated on both the inside and outside of the flanges and on the barrel. Although the round tumbler is used successfully with the ordinary type of bucket, it is considered preferable to widen the bottom of the bucket somewhat when a tumbler of this type is employed. This usually is accomplished by placing a flange along the outside of the bucket at the bottom between the front and rear eyes.

Screen plates are also being successfully made from manganese steel. When used in connection with gold dredges, the hole may be either $\frac{1}{2}$ in. in diameter, tapering to $\frac{5}{8}$ in. on the outside, or $\frac{5}{8}$ in. in diameter, tapering to $\frac{3}{4}$ in. on the outside. In order to overcome the difficulties of casting round holes in manganese steel, some plates have been designed with rectangular slots about $\frac{1}{2}$ in. or $\frac{5}{8}$ in. wide on the inside, tapering to $\frac{5}{8}$ or $\frac{3}{4}$ in. on the outside, the slots being $1\frac{1}{4}$ in. long and set diagonally. Elevator buckets of the same thickness as malleable iron buckets are now made of manganese steel, the latter giving six times the service of the malleable iron.

The application of manganese steel to steam-shovel dippers was one of the first problems attacked by the manufacturers of this material in this country, the wear in this service being particularly severe. A dipper known as the Missabe type is made entirely of manganese steel. Its body consists of but two castings, the front and back halves, which give maximum stiffness and rigidity. This design permits the construction

of a large dipper with only a few rivets. The bail brackets are set at an angle conforming to the line of pull on the bail. They are butted against shoulders or offsets which are formed integrally with the sides of the front castings and are riveted to the front half of the dipper, the joints between the front and back halves being placed at the rear of the brackets, thus throwing all the digging strains on the front casting, and relieving the



FIG. 7.—FIVE CUBIC YARD MISSABE TYPE MANGANESE-STEEL DIPPER WITH VANDERHOEF FRONT.

rivets at the joints from shearing stresses. The teeth, which are easily renewable, are of cast manganese steel.

Dippers of the Missabe type of 10-cu. yd. capacity, which are believed to be the largest manganese-steel dippers ever made, were used on dredges working in the Culebra Cut, Panama Canal. The weight of the empty dipper is 37,600 lb. and its overall dimensions are 9 by 9 by $10\frac{1}{2}$ ft. The lips are $3\frac{1}{4}$ in. thick and the fronts are the same thickness underneath the teeth. Between the teeth the fronts are $1\frac{1}{2}$ in. thick, the thickness

at the bottom band being $3\frac{1}{4}$ in. The backs are $1\frac{1}{4}$ in. thick, except at the bottom where the thickness is increased to $3\frac{1}{4}$ in.

Another improved dipper front is known as the "Vanderhoef," in which the front and the dipper-tooth bases are cast integrally. The renewable points are substantially designed and are of the reversible type. Among the advantages of this design may be mentioned the elimination of rivets, correct digging angle, increased room inside the dipper, due to the position of the base portions, great strength coupled with rigidity of the entire front, and economy in repairs and renewals.



FIG. 8.—BLACK TYPE VENTILATED HEAD FOR NO. 21 GATES CRUSHER.

Fig. 7 shows in detail a Missabe type manganese-steel dipper of 5-cu. yd. capacity, equipped with a Vanderhoef front.

Fig. 8 shows a cast manganese-steel ventilated head of the Black type, suitable for a No. 21 Gates gyratory crusher. This casting weighs 21,360 lb. As shown in the illustration, it is provided with a series of vertical flues or cores running from top to bottom which permit a circulation of air between the wearing surface and the shaft. This tends to prevent the transmission, from the wearing surface to the shaft, of the heat incident to crushing. As compared with a solid head, the weight is reduced from 20 to 30 per cent.

Manganese steel, although not suitable for railway car wheels on

account of its comparative softness, has been found satisfactory for mine- and skip-car and crane wheels. For crane wheels carrying heavy rolling loads, manganese steel has been found particularly satisfactory. Manganese-steel crane wheels are in service today carrying loads of 90,000 lb. each, although 30,000 lb. has always been considered the limit of safe working load for chilled-iron wheels. A microscopic examination of a manganese-steel wheel under load will show a flat spot instead of a line contact as in the case of hard-metal wheels; manganese steel under these conditions is springy and recoils as the load is released. The wheels also exhibit strong resistance to flange wear. This feature is worthy of consideration in view of the fact that derailments on curves are usually caused by the flanges having worn to the angle of least frictional resistance, allowing the inherent straight line force to push the wheel flange over the rail.

Where the loads are heavy, manganese steel is now being extensively employed for gears. For the past two years manganese-steel spur gears have been made with the teeth ground mechanically true, up to 44-in. pitch diameter with a 10-in. width of face. Many ordinary steel gears are cut with a range cutter which does not leave the rolling contact theoretically correct, whereas in grinding manganese-steel gears, a trimming arrangement is placed on the grinder for the purpose of dressing the emery wheel which shapes the tooth to the proper contour.

After two years' continuous service, the wear on a set of heavy high-speed manganese-steel nest gears was recently measured. These gears had a 10-in. face and when new the thickness of the tooth at the pitch line was 0.684 in. At the conclusion of the two-year period, the tooth measured 0.601 in., the wear being 0.083 in. These gears are still in service after having more than earned their original cost, and barring accident they will be good for many years to come.

Manganese steel is now employed extensively in rolling-mill machinery; it has been found particularly suitable for pinions on heavy roughing and blooming mills. Other parts around steel mills now being made of manganese steel are pipe-drawing balls, chafing plates, sprockets, cams, draw chains, unloader chains, conveyor chains, buckets, screens, pulverizer hammers, dry and wet pan parts, crusher castings, coupling boxes, spindles, and backing rolls.

For the production of manganese-steel castings the shop practice is fundamentally similar to that of an ordinary steel foundry, but a number of factors not vital in ordinary work must be considered. Among other points, the shrinkage is $\frac{5}{16}$ in. per foot as against $\frac{3}{16}$ to $\frac{1}{4}$ in. for ordinary steel. For this reason the founders frequently prefer to make their own patterns from drawings supplied by their customers. Where patterns are furnished, sometimes they have to be modified or rebuilt.

The distribution of the metal in the casting is also important, since

the heat treatment and cooling produce stresses which must be taken care of. At times it is necessary to add metal to a casting temporarily to withstand shrinkage strains, such additions of metal being removed in the machine shop. Manganese steel, being comparatively expensive, must be used economically, for which reason designs or patterns occasionally have to be altered.

In making manganese steel, side-blow converters are commonly employed and the 80 per cent. ferromanganese with which the steel is alloyed is melted in separate crucible furnaces. The melted alloy is placed in the bottom of the ladle and the converter metal is poured in on the top.

Manganese acts as a strong scavenger, removing the gases and impurities from the metal, leaving it condensed and homogeneous. Before the molds are poured, the ladles are allowed to stand for a time to permit the alloy to perform its function, after which the accumulation of slag is removed.

The annealing process is particularly important. The temperatures and the time the work is allowed to remain in the furnace vary considerably, depending upon the nature of the piece; the temperature ranges from 1,600 to 2,200°F. and the time of annealing from 4 to 24 hr.

The gates and risers of castings can be broken off as in malleable foundries. The cores are removed with pneumatic drills and the exterior surfaces are cleaned and trimmed by grinding.

Castings which are to be accurate are finished in the machine shop by grinding, much of the work requiring special grinding equipment. Holes more than $1\frac{1}{4}$ in. in diameter are cored out of the casting and ground to size. When it is necessary to drill smaller holes or to cut threads, soft steel or wrought-iron inserts are set in the molds at the desired points, like chaplets, and the metal is cast around them. Except in small holes, inserts or bushings are not necessary where keyways are cut, since these can now be ground successfully. Sometimes bushings are set in the hubs of wheels to be machined by ordinary methods.

The foreman continually exercises his judgment as to the wheels and cuts to be employed; coarse wheels and heavy cuts are used for removing stock rapidly, though it is never accomplished as easily as would be the case were it possible to use ordinary cutting tools.

A complete manganese-steel castings plant, therefore, includes a steel foundry with special equipment for melting ferromanganese and annealing castings, a pattern shop, and a machine shop equipped with heavy-duty standard and special grinding apparatus.

Protecting California Oil Fields from Damage by Infiltrating Water

Discussion of the paper of R. P. McLAUGHLIN, presented at the San Francisco meeting, September, 1915, and printed in *Bulletin* No. 108, December, 1915, pp. 2313 to 2319.

M. E. LOMBARDI, Berkeley, Cal.—Mr. Bell, I would ask you to state before the meeting what you said to me the other day, about the final recovery of oil from oil sands that have become inundated with water.

A. F. L. BELL, San Francisco, Cal.—In stating my opinion, I give it with certain qualifications. In the Kern field, and possibly in some others, I think we will eventually get all the oil out, whether the water comes in or not, since it is peculiarly constructed, being in the form of a partial dome. In 1909, while I was field manager of the Associated Oil Co., I installed some large air compressors to handle the water. At that time the oil had migrated to the higher and more shallow portions of the field, because the water had inundated the deeper sections. The result was that the shallow districts within the center of the dome were in many cases producing as much or more oil per well than the deeper wells. And in the inundated sections the water had slowly spread over large areas, so that nothing but water was pumped from them. Under these conditions, we decided to pump the wet well with air, and installed four 2,000-cu. ft. compressors.

The air was compressed and forced into the wet wells at about 250 lb. pressure. Shortly the inundated district that had produced almost all water and only about 150 bbl. of oil a day increased its production to over 2,000 bbl. of oil per day. It was not long before we noticed that the production of the shallow territory was falling off to such an extent that in many cases the wells were abandoned; at the same time the area of the water-inundated district was greatly reduced, showing that when the water pressure was reduced the oil migrated back to the lower depth. The wells which at first had had the air jets put into them finally produced so much oil that we had to take the air jets out and return to the use of the beam pump, because it is not commercially feasible to pump an oil of 14° gravity with air. To handle such oil profitably by air, it must carry with it 50 to 90 per cent. of water.

The results have led me to believe that on monoclines, where there is a gradual rise to a certain point, or in plunging anticlines, even though water gets into oil sands, eventually we get the larger percentage of the oil. However, the man with a small acreage in the deeper sections may find some morning that his oil has gone, another man in some higher portion of the district having gotten it.

I do not use this as an argument against any of the means adopted for the prevention of the inundation of water, since infiltration is always

a source of danger. Many of my theories have not always worked out, and this is only a theory of mine.

A. C. McLAUGHLIN, San Francisco, Cal.—Supplementing what Mr. Bell has said, the division ownership of lands makes necessary some solution of the water question. Mr. Bell's theory is undoubtedly correct in some cases. However, if there are some 25 or 30 owners, each would like to get the original amount of oil which underlaid his lands.

A. F. L. BELL.—My theory is that to get oil out of an inundated district, you must reduce the liquid level so that water and oil flow together to the well; but if the water level cannot be reduced and remains at a high level in the well, then the water is a detriment and will shut off the oil. To get all the oil you must have the water under such control that you can pump more water and oil from the well than is coming to it. The moment more water comes to a well than can be handled, then the oil is shut off.

A. C. McLAUGHLIN.—By reason of the higher surface tension of water over oil, the tendency is for the water to migrate to lower pressure areas; where pressures are practically nil, the conditions are as Mr. Bell mentions them, but in cases where there are pressures of 25 or 30 lb., the tendency is to migrate toward the well. This displaces the oil and creates an area of low pressure when a well is dug. So I think it is conceivable that by letting water into oil wells, the oil could never be gotten out.

WILLIAM A. WILLIAMS, San Francisco, Cal.—The Bureau of Mines is charged with the supervision of operations on Indian lands in Oklahoma, and we have had the water menace to contend with on these lands. The Cushing field is perhaps one of the best illustrations of the damage that may be done to a field by not effectively excluding the water from the oil and gas sands. On March 1, 1915, the production of the Cushing field was in excess of 300,000 bbl. per day; today it is less than 140,000 bbl. The excessive decline is due to improper methods of casing wells to shut off the water. The operators did not case off the water uniformly; some cased above a certain sand and some cased below the same sand. As there are several producing sands, both oil and gas, in the Cushing field, the result of this unsystematic casing has been intercommunication between sands, permitting the gas to come into contact with the water-bearing sand, and while the gas pressure was sufficient to overcome the hydrostatic pressure of the water sands, the gas forced the water away from the bore hole. When this gas pressure was not sufficient to overcome the hydrostatic pressure, the water found its way into the oil sands, resulting in an abnormal decline in production and the formation of a considerable percentage of cut oil or emulsion.

By the drowning and wasting of gas a considerable loss occurred not only in the value of gas as a fuel, but in the matter of operating costs, for where gas occurs with the oil, this gas, when properly conserved, assists in pumping the wells, thus reducing operating costs.

It was found impracticable to protect with a string of casing, each sand found in the Cushing field and in lieu of the large amount of casing necessary to protect the sand adequately, the Bureau of Mines has recommended sealing with mud all oil, gas, water, and barren sands, above the productive sands. This is accomplished by landing a string of casing immediately above the sand to be utilized, sealing with mud all sands behind this casing.

In one of the Kansas fields, where the gas occurred at 1,500 ft. and oil at 2,500 ft., a strong flow of top water was encountered above the gas sand. Wells were drilled, the water and gas sands mudded off behind a single string of casing; the oil wells were drilled in at a cost \$500 less than required to complete wells without the use of the mud fluid, notwithstanding that the well had to be drilled for a considerable distance with the fluid in the hole. The saving was accomplished in the reduced amount of casing required, the company drilling separate wells for gas.

A. F. L. BELL.—A law should be enacted requiring that all sands above the oil-producing sand should be cased off.

WILLIAM A. WILLIAMS.—It is necessary that all water sands, in fact all sands, be cased off uniformly. Unless this is done, water will find its way through the sands into the producing sand notwithstanding the fact that water may have been effectively shut off at different casing points in the various wells.

MARK L. REQUA, San Francisco, Cal.—I would like to ask Mr. McLaughlin whether he has decided how far he can go in remedying the water situation. I believe the conditions in certain parts of the State have gone to such an extent that it is dubious whether they can be remedied.

R. P. McLAUGHLIN.—Many old wells have been drilled with no records to show what was put in them and what the conditions were originally. Many of those old wells we cannot remedy since it might cost more to fix them than the wells are worth.

This would be a mere guess, but out of all the old wells we would be fortunate if we could at present fix up 10 or 15 per cent. However, in say, 90 per cent. of the new wells, we should be able to give the operators advice so that we could entirely avoid the damage. One of the largest fields in the world, the Midway-Sunset, has now only about one well to 30 acres of proved oil land, which gives an idea of the great importance of wells yet to be drilled.

MARK L. REQUA.—I am in entire accord with this work, but are we not, in a measure, shutting the door after the horse has run out of the stable. It is true, so far as new wells are concerned, that it will be possible to do much under the supervision of the State Mining Bureau, and in a large measure to avoid the unsatisfactory conditions which exist at a lot of wells already drilled, but in many old wells it will be impossible to shut off the water.

I have a case of that kind in the Coalinga field, where we drilled a pioneer well. We did not know the formation at the bottom and drilled into an underlying water sand which was not recognized at the time. The hole was not plugged; the casing collapsed, and we redrilled the well. We do not now know where the old casing is. It is within a reasonable limit of the bottom of that hole, but we are utterly unable to get down over the exact spot, clean the hole, and plug it.

That well is producing water, the old casing allowing the entrance of water from the underlying strata.

WILLIAM A. WILLIAMS.—The water might be shut off by cleaning out the well and filling up with mud, if the well is to be abandoned. In Illinois they have had trouble with edge water. When edge wells begin to show water, the operators cement up 10 or 15 ft. of the bottom of the sand, effectively shutting off the water until the water level rises above the top of the cement. The well is again cemented and so on until all of the oil has been exhausted from the sand, when the sand should be effectively plugged, otherwise water may get in and pass through the upper sands into the producing wells. The method of cementing was simple but effective, consisting of placing the necessary cement in the bottom of the well, forcing the cement back into the sand with a pressure equal to a column of water possibly 100 ft. above the ground-water level.

C. D. KEEN, Shreveport, La.—In Louisiana we have no laws which deal with the shutting off of water from the oil sand. About two years ago the Governor of the State appointed a committee of representative oil men from all parts of the State, and requested them to formulate a set of regulations dealing with the conservation of oil and natural gas.

This committee submitted its report, which passed through the legislature, and is known now as the Conservation Law of the State of Louisiana. I was a member of that committee. The point was brought up, whether it would be necessary to embody in the regulations a section, making it compulsory for the oil operator to shut off the salt water from the oil sand.

After some discussion it was the belief of the meeting that the successful shutting off of the water from the oil-bearing strata is of such vital

importance for the oil operator that it would not be necessary to make any regulations covering that subject. It is impossible in our part of the country to complete an oil well unless the water from the strata above has been shut off entirely.

It may be interesting to give you a short description of the methods we use to shut off the salt water.

In our wells we set a string of 6-in. casing at a depth ranging from 2,200 to 2,500 ft. After we have drilled the hole down to the point where we wish to set the 6-in. casing, we put the casing into the hole; when it reaches the bottom of the hole we displace the drilling mud inside the 6-in. casing by running about 500 ft. of plugged 4-in. drill pipe inside it. We then mix our cement, usually 30 sacks of cement and 15 sacks of sand, and pour the cement on the heavy mud in the 6-in. casing. When the cement is poured into the 6-in. casing we follow it up with a wooden plug about 3 ft. long and $5\frac{1}{2}$ in. in diameter, provided with rubber washers on both sides, so as to make a tight fit in the casing. After the plug is driven into the pipe, the swivel is screwed on and the string of pipe is lifted up about 8 in. from the bottom; then the pump is started, and the cement with the plug behind it goes to the bottom of the hole. The pump keeps on running until all the cement is pumped outside the bottom of the 6-in. casing, and the plug hits the bottom; this shuts down the pump, and we know that there is no more cement left inside the 6-in. casing. In order to make sure that the job has been a good one, a steel measuring line is run into the casing to see if the wooden plug is actually down on bottom.

We allow the cement 10 days to harden, and then bail the casing dry. If the cementing has not been successful, we cement the 6-in. over again, or set a string of $4\frac{1}{2}$ -in. casing inside the 6-in., but under no circumstance is the well completed unless the water is shut off satisfactorily. The shutting off of the water in Louisiana has not given much trouble. Until about three years ago we had to rely on gumbo formations to set our casing in, but with the cement process we can set the casing in almost any formation to shut off the water.

I find that the process of cementing in California is somewhat more complicated than ours, which may be explained by the fact that the casings are bigger than ours. It is well known that it is easier to cement a string of 6-in. casing than a string of 12-in. casing, and I doubt very much if cementing with the single plug would be successful in California.

The committee mentioned above made one regulation which I would like to mention. The surface formations in Texas and Louisiana are soft and unconsolidated. In the new fields we meet with gas pressures ranging from 450 to 800 lb., and the blow-outs we have are sometimes serious. In different places blow-outs took place, forming craters in less than 24 hr., in which the derrick and all machinery were buried; one of

the most notable instances of that kind was the burning gas well close to Oil City, La., where a crater was formed about 250 ft. in diameter and about 50 ft. deep. In order to deal successfully with conditions of that kind it was deemed necessary to set in each well a sufficient amount of surface casing, approximately 200 or 300 ft., and cement it. When the surface casing is cemented in good shape, and a blow-out is encountered, one has at least one good string to rely on, and the operator is in good shape to kill the blow-out.

In connection with my remark about the use of big casings I would like to ask why such large casings are used in California. A few days ago I visited the Coalinga field, and was told of a well in which was set 2,800 ft. of 12-in. casing. The well was drilled all the way down to the casing point with the rotary system. Would not 8-in. casing have served equally as well?

In our Gulf Coast fields, we get wells in which the casing that shuts off the water is not bigger than 6-in.; and we get some good wells, which make all their production through 6-in. casing and 4½-in. liner.

In my opinion, to use a smaller casing in California would save both time and money.

M. E. LOMBARDI.—We always try to shut off with 10 in., where the collapsing pressure which the pipe will have to bear does not require too heavy a pipe. The object is to finish with 6⅝ or 6¼ in. at least, as we find a hole less than 6¼ in. an expensive one from which to produce.

We have to use such small holes, at times. In the western side of the Coalinga field we shut off with 10 in. and finish with 8¼. But a condition obtains there (I do not know whether it is edge water coming in, or what), that often requires a second shutting off. If it is first shut off with 8 in., the second requires 6¼, and next 4½, and that is not a proper or economical well.

A. C. McLAUGHLIN.—What percentage of the cementing operation is not successful?

C. D. KEEN.—The majority are successful. I have cemented about 45 wells, and I lost only one cementing job on account of some gas escaping.

WILLIAM A. WILLIAMS.—What kind of set shoes do you use in Louisiana?

C. D. KEEN.—We use a light shoe, about 2 ft. long. Some operators only use reinforced couplings. We do not use heavy shoes at all.

A. F. L. BELL.—Answering Mr. Keen's question, why we use large casing, I might say that in California the conditions are so different from those in the East that he is not the first or only Eastern man who

has thought we were extravagant in our work; but there are conditions arising which cannot be understood or appreciated by anyone who has not operated in California fields. Many wells in the East are drilled and placed on the pump for what it costs us to build and equip the derrick and rig on our deep wells. The latter cost us about \$4,000, and it would be the height of folly to erect them for less. We have formations entirely unknown in the East, consisting of caving ground, and running sands. These are so different from the solid sandstone and limestones in the East that in our experience we have found it was a good plan to use larger pipes and more of them than would be necessary in many other districts.

C. D. KEEN.—In our company we finish all our wells with the rotary. We drill to the oil sands with thin mud, set the liner in the well, and wash it inside and outside with clear water.

In California, the opinion is that the rotary mud will hurt the well by filling up the voids in the sand. If there is the least gas pressure present to bring the water back out of the formation, it does not hurt the well at all to drill with a rotary.

A. C. McLAUGHLIN.—With rotary drills some wells in California have been drilled in which the oil sand was passed entirely through without its being recognized. The rotary is not satisfactory unless the depth of the oil sand is known before drilling begins.

A. F. L. BELL.—In California our formations do not lie as level as those in the East. In fields that I am familiar with in the East you can safely say that the drillers are going to strike the oil sand in a few feet. In California the conditions are entirely different. There are so many underlying dips and folds that extreme care must be taken in locating the sand in many of the fields. In one case, in the Midway field, we drilled two wells with the rotary at the same time, and in each case we passed through the oil sand without knowing it, and the only thing noticeable while drilling was a few bubbles of gas at one point. In looking for the oil we drilled down into water and at first thought the district was barren, as there was not the slightest indication of oil. Then, after studying the conditions, we thought that possibly the oil sand might be where the little ebullition of gas had been noted. We then cemented off the bottom water in each well; one well came in at 10,000 bbl., and the other at 5,000 bbl. In still another case the rotary showed great quantities of oil in the mud and we expected a gusher, and got a 15-bbl. well. We decided to stop the rotary drilling above the oil sand and finish the well with standard tools.

Petroleum as Fuel under Boilers and in Furnaces

Discussion of the paper of W. N. BEST, presented at the San Francisco meeting, September, 1915, and printed in *Bulletin* No. 104, August, 1915, pp. 1527 to 1537.

WILLIAM A. WILLIAMS, San Francisco, Cal.—I would like to ask whether Mr. Best ever had any experience with an oil burner that used neither air nor steam?

W. N. BEST, New York, N. Y.—Yes. This type is commonly known as the "mechanical spray burner" and uses oil heated under high pressure to above its vaporizing point. These burners are not as efficient as the atomizing type for the following reasons:

1. They produce a round flame which, as with any other type of burner, must pass over a flat surface. A round flame cannot fit a flat surface.

2. They do not evenly distribute the heat, resulting in localized heat.

3. The outlet through which the fuel passes is very small; so all gravities of oil, especially oil containing sand, cannot be burned.

4. Their construction is such that they must be cleaned frequently, as the residuum in the fuel clogs the burner.

5. The oil opening being small, the capacity of each burner is small, so that many burners must be used in a boiler or furnace. (The fewer burners there are, the more accurately they can be regulated.)

In marine boilers, these burners are largely used, because they do not require either steam or air for atomization. However, there are other methods by which this objection to other types can be eliminated, such as by the use of a rotary blower and steam turbine engine, the exhaust from the engine passing into the engine condenser, thus obviating loss of water.

Mechanical burners are rarely used in shops. I know of but one power plant using them.

With mechanical burners, the fuel must be maintained at a high pressure. On board naval vessels, some mechanical burners demand an oil pressure of from 100 to 600 lb. It is advisable to reduce the pressure as much as possible, as fractured pipes or fittings are particularly dangerous, for the heated oil escapes as a vapor. In such cases there is liability of an explosion caused by spontaneous combustion.

HENNEN JENNINGS, Washington, D. C.—Is crude petroleum required for use under boilers, or is it possible to extract certain constituents of petroleum and then use the residuum under boilers? Is it not, with the limited production in the country, necessary to make a full use of petroleum before using it as a finality in producing steam?

W. N. BEST.—I have always contended that any burner which will not atomize any gravity of topped oil, crude oil of any gravity, fuel oil, oil tar, or coal tar is not worthy of the name burner. We have boilers equipped using coal tar as fuel, others oil tar, others crude oil as low as 11° Bé. gravity (Mexican oil), while some plants use oil containing as high as 3 or 4 per cent. sand; in fact all kinds of liquid fuel.

DAVID T. DAY, Washington, D. C.—What is the size of this 3 or 4 per cent. of sand, and what amount of dirt can go through your process?

W. N. BEST.—In the Mexican fields the oil is stored in large reservoirs or lakes and in pumping it up into tanks you will often find that it contains some sand of average size; therefore, a burner must have the oil opening of sufficient size to allow this sand to pass out readily. The oil opening should be independent of the steam or air atomizing opening so that the force of the steam or air will not cause the sand or other residuum in the fuel to cut away the metal of the burner. For this reason the exterior atomizing type of burner is always preferable. The average mechanical burner has an opening about $\frac{3}{64}$ -in. in diameter and any sand in the fuel must of necessity cut away the metal in the opening of the burner or sprayer.

In southern California, some oils contain as much as 2 per cent. sand, so fine that it cannot be separated from the oil by the use of strainers. With some types of burners the mouth is cut away by this fine sand. In such burners it is necessary to have removable parts in order to save the body of the burner. It is always advisable to have an abundant supply of repair parts on hand for an internal atomizing burner.

We often burn coal tar containing as much as 30 per cent. free carbon; the burner must be so constructed that the free carbon can readily pass through it.

MARK L. REQUA, San Francisco, Cal.—Do you know of any burner adapted to domestic use which can be installed at comparatively moderate cost? Burners are being used in apartment houses, hotels and residences, but they cost so much that it is not feasible to put them into small houses. There is a field for a small burner if installed cheaply, that would supplant the use of coal. Do you know of any burner of that type?

W. N. BEST.—There are a number of air carbureting burners which require no mechanical means for the atomization of the oil. The oil must be volatile (a distillate oil) so that it will readily burn on a hot plate.

In this hot plate are a number of openings to provide the requisite quantity of oxygen for burning the oil. And around the plate there is a refractory lining, the heat therefrom aiding combustion.

These burners can be installed very cheaply, the only difficulty being that they cannot be installed in some States in dwellings, etc., because of the fact that they ordinarily use gravity feed. A large number using gravity feed are in use in southern California, and in the Eastern States they are being used in small boilers for heating green houses and the like.

MARK L. REQUA.—There is a hot plate which uses hot water and oil.

W. N. BEST.—The trouble with this type of burner is that the oil used is not sufficiently volatile. A very volatile oil, a distillate, is required. Again, oil and water do not mix well and the quantity of water in this type of equipment must be accurately controlled. If anything goes wrong with the oil supply, too much water is fed and the apparatus is flooded.

A. F. L. BELL.—I would like to answer Mr. Jennings' question, as I do not think it was fully covered. Mr. Jennings wanted to know whether all petroleum should not be topped before using it. Practically all light petroleum in California is topped, as about one-half the fuel oil consumed in California is residuum oil; the other half is crude oil, too heavy for refining purposes.

M. L. REQUA.—I think Mr. Jennings has in mind the idea that to burn any oil before it is subjected to some refining process is not an economical way.

As far as the product resulting is concerned, the final residuum is as good for fuel purposes as crude oil, and it has practically the same number of heat units, or, in fact, a few more. And it does remove a certain economic waste which exists in burning the crude oil before it has been treated. The sooner the oil operators realize that they are wasting resources which should be utilized, the better it will be, I think, for all concerned.

Sliding Royalties for Oil and Gas Wells

Discussion of the paper of ROSWELL H. JOHNSON, presented at the San Francisco meeting, September, 1915, and printed in *Bulletin* No. 102, June, 1915, pp. 1291 to 1294.

WILLIAM A. WILLIAMS, San Francisco, Cal.—A sliding scale of royalties was proposed to the Department of the Interior in connection with the fixing of royalties to be charged upon oil leases made on the Osage Reservation in Oklahoma. The possibility of evasion and difficulty of enforcement required close inspection to satisfy all interests, particularly in connection with the exemption from royalty on oil to equal maintenance and interest on the junk.

Mr. Johnson discusses the block method, the period method, and the cumulative method, and favors the adoption of the cumulative method with a proviso that the wells should be permitted to be pumped until the gross income is as low as the maintenance cost, which would mean exemption from royalty on the smallest wells. He recommends that depth be the variable factor upon which the sliding royalty is based and suggests that on partially developed territory in the Osage, the royalty be one-sixth for oil wells averaging less than 2,000 ft. in depth, one-eighth for wells from 2,000 to 3,000 ft., and one-twelfth above 3,000 ft.; for the rest of the Osage he suggests one-eighth above 2,800 ft. and one-twelfth below that depth. The net result, if Mr. Johnson's suggestions had been accepted in the Osage, would have been to reduce the then existing rate of royalty and considerably increase the cost of administration.

The difficulties in determining the size of wells to be exempt and the possibility of obtaining oil at different depths in the same well might lead to complications which are avoided by charging a flat royalty as now universally charged throughout the United States. It is not uncommon to reduce the royalty when the production of the wells becomes unprofitable at an existing rate of royalty, accomplishing very much the same results as Mr. Johnson has attempted to accomplish by the cumulative royalties, and probably in a manner far more satisfactory to the operator and the lessor.

A. C. McLAUGHLIN, San Francisco, Cal.—The net revenue of the well depends, first, on the cost of production; second, on the amount of oil produced; therefore it would be impossible under Mr. Johnson's system to set a royalty which would be fair to all concerned. If it is impossible to set it at the beginning, it is certainly further impossible to do it from day to day. I think it is entirely impracticable of application.

WILLIAM A. WILLIAMS.—Mr. Johnson said this would be based on pipe-line statements of the runs.

A. C. McLAUGHLIN.—But the price obtained for the oil and the cost of production are big factors. I know of some wells with an average production of 150 bbl. a day, and costing to operate say, \$150 a month, each; but when they get to be strippers, the operating costs are \$25 a month. It is impossible to set an equitable rate of royalty on that basis.

A. F. L. BELL, San Francisco, Cal.—It seems to me that on Government lands in California a sliding royalty should be a dangerous thing to inaugurate in the oil business. I think there should be a period royalty.

I made an examination of about 500 wells, in one of the Oklahoma fields, and found the average life to be about 7 years, after which the wells produced less than a barrel per day.

In California the production of the well would be greater and the profitable life larger. Whatever we do let us try and keep away from a sliding royalty since it would add to the already heavy burden of the oil companies.

ROSWELL H. JOHNSON (communication to the Secretary*).—The paper consists of (a) an analysis of the different methods of graduating royalty, (b) an examination of each as applied to oil wells, (c) two positive recommendations.

Has not Mr. Bell confused the complexity of the types of sliding (which I mentioned only to reject for that very reason) with the two kinds I favored?

I have recommended that a different rate be used in fields where costs are light and receipts heavy than that used where costs are heavy and receipts light. This is merely to yield to what is already a tendency, and what is very general in nearly all businesses that have royalties; the royalties of authors, lecturers, playwrights, mine owners and others vary according to just such considerations of costs and receipts. It is more scientific to vary the royalty than to have a universal fixed royalty and varying bonuses, because lower royalties in expensive regions are necessary to make developments in such regions possible.

My other recommendation is that a definite number of barrels (in some fields it would be one) be exempted from all royalty payment in order that the life of the well may be continued. This seems to me to be extremely simple as compared to many sliding arrangements that are in widespread use. In fact, it is one of the simplest and easiest that can be devised, and gradually lowers the royalty rate step by step as the well declines. It is, technically, a truly "degressive" rate.

Mr. McLaughlin very properly contends that I should have specified

* Received Oct. 25, 1915.

the time in the history of the well at which the cost of maintenance, etc., is determined. That time is when the well is old and nearing abandonment, since the main purpose is to prolong the life of the well. The fact that maintenance is higher per well earlier need not concern us.

The contention that late maintenance is variable is unimportant also, since we cannot hope entirely to prevent unduly expensive wells from being abandoned earlier. The amount should be fixed at the nearest even number of barrels above the cost of maintenance and "interest on junk" of the *average* old wells in that district. To be sure, price per barrel changes, but when it is higher, costs are in general higher, so that it is in some degree compensatory.

In regard to the Osage wells, the fact that at times some wells operate at less than capacity need not interfere in any way, since there is all the more reason to try to assist in keeping small wells alive during such times of stress, as it is at such times that we get most of the premature abandonments of wells that are needed later.

I see no difficulty in the way of enforcement, for we simply subtract a constant number from the pipe-line runs before computing the royalty. The well need not be seen.

There is on the other hand an administrative feature in the classification of well by depth (or other element in the cost of receipts). However, the depths of the several sands are so well known that in very few cases would the operator dare to return a false statement, even should he be inclined to do so.

The government in insisting on one-sixth royalty in the Osage, no matter how small, old, or deep the wells on a property are, seriously offends against conservation. It should be urged to consent to a reduction of royalty in the leases now existing when their wells are near abandonment, and to avoid this unfortunate situation in the new leases soon to be bid for by making express exemption from all royalty of a barrel a day. The Indians would be compensated by the higher bonuses that would be bid.

Important Topping Plants of California

Discussion of the paper of ARTHUR F. L. BELL, presented at the San Francisco meeting, September, 1915, and printed in *Bulletin* No. 105, September, 1915. pp 1769 to 1799.

WILLIAM A. WILLIAMS, San Francisco, Cal.—Is the Union Avila plant run as a topping plant or as a dehydrating plant?

A. F. L. BELL, San Francisco, Cal.—Both. It takes off an average of 20 per cent. water besides topping the oil. Part of the crude oil is run through an electrical separator before running to the topping plant, but the entire crude averages about 20 per cent. when it enters the topping plant.

DAVID T. DAY, Washington, D. C.—I suppose most of the work contemplated in these large topping plants is simply to deal with topping the oils with only a small percentage of water in them.

A. F. L. BELL.—Not more than 2 per cent. of water can be handled in ordinary stills without causing trouble. At our Gaviota refinery, where the oil runs from 15 to 30 per cent. water, we carry 20 lb. pressure on our stills to hold the framing down, we can do that with small stills with large ones it would not be safe; but with a retort, the quantity of water contained in the oil does not affect its working. It simply means additional fuel to evaporate the water.

DAVID T. DAY.—In your judgment is it the proper thing to dehydrate and top the oil at one operation; or do you think it would be better to dehydrate with the electrical appliances first and then top afterward?

A. F. L. BELL.—Where the water in the crude exceeded 22 per cent. I would recommend dehydrating first by the electrical, or some other process that did not necessitate evaporating the water.

DAVID T. DAY.—Would that apply to oils mixed with fresh water as well as oils mixed with salt water?

A. F. L. BELL.—Yes, with any water. The cost of fuel for refining increases in proportion to the quantity of water in the oil. The water should be separated, if possible, before it goes to the stills; but with less than 2 per cent. water such separation would not pay.

The Cost of Maintaining Production in California Oil Fields

Discussion of the paper of M. E. LOMBARDI, presented at the San Francisco meeting, September, 1915, and printed in *Bulletin* No. 105, September, 1915, pp. 2109 to 2114.

C. D. KEEN, Shreveport, La.—Did Mr. Lombardi include in his figures the purchase price of the property?

M. E. LOMBARDI, Berkeley, Cal.—No.

C. D. KEEN.—Suppose I pay \$1,000 per acre for it, and in four or five years it is worked out.

M. E. LOMBARDI.—That is a question of depreciation and must be added to my figures.

MARK L. REQUA, San Francisco, Cal.—You say it cost \$35,000 to \$40,000 a year to maintain 1,000-bbl. per day production?

M. E. LOMBARDI.—Yes, 1,000 bbl. a day would cost \$40,000 a year to maintain.

M. L. REQUA.—It is rather a coincidence that the figures in my mind for some time check almost exactly with those Mr. Lombardi has arrived at, after examining the data which he has had at his command. I think he has made a conservative estimate. It is peculiarly opportune, as the average Californian has not considered what it costs to produce oil in this State. Interest on the investment, depreciation and amortization, and such matters as that have never been taken into consideration. The popular notion is that it cost so much to pump oil from the wells. The indirect costs are more than the costs of operating the wells and pumping the oil out of the ground, and that is the reason the industry is depressed at the present time. Taking the industry over the State, nobody is making money, and that is because it has not been realized what it costs to produce oil. With oil selling at 35c. to 45c. a barrel, there is no profit for the producer.

H. M. BACON, San Francisco, Cal.—I checked that up on one of the leases under my charge running over a period of three years. Upon completion of our second well the development cost of oil produced was 67½c. The average price of all the oil sold at that time was about 57½c. As the development was continued and more wells were brought in, these figures about balanced, but with oil selling at the price obtaining today, considering operating and interest charges, the investment shows little or no profit to date.

Discussion of the paper of L. G. HUNTLEY, presented at the San Francisco meeting, September, 1915, and printed in *Bulletin* No. 102, June, 1915, pp. 1333 to 1349.

E. W. SHAW, Washington, D. C. (communication to the Secretary*).
 —The comparatively great extent and uniform character of the Dakota sandstone make it of great interest to the student of rock pores and the movements of their fluid contents. However, compared with other sands in areas of carbonaceous rocks facts concerning it do not seem to be of such great direct importance to the producer for it has thus far yielded few important pools of oil or gas. Information concerning the sandstone is extensive but scattered, hence a general treatise regarding it—particularly its pores, the fluids they contain, and the movements of the fluids—has been much needed. I therefore feel that the chief thing to be desired concerning Mr. Huntley's paper is that it were more extended, particularly as to the discussion of the data presented. The general structure map of the Dakota is of especial interest.

Slight changes in wording seem to be desirable here and there in the paper. For example, at the beginning of the third paragraph one wonders if flora instead of fauna were not intended. In Figs. 2, 3, 4, and 5 the vertical scale seems incorrect. Concerning the analyses of oil quoted it is noted that they "are given, as indicating what may be expected in different localities." This seems to need modification for the average analysis of oil from Powder River, which indicates for example that this oil begins to boil at 213°C. The fact is that the samples to which this figure refers were taken from seeps long exposed to the air and hence the underground Dakota oil of this district no doubt differs profoundly from that which the analysis indicates. The product of a well here would probably differ more in boiling point and other respects from the samples analyzed than it would from almost any other petroleum in the world. The use of the word synclinorium does not conform to the original definition, but this may be pardonable because misuse of the term, though a different misuse, has become general. Synclinorium is an orographic term and cannot be correctly applied outside of mountain regions, for it is derived in part from the Greek word for mountain. It has, however, been widely used in a purely structural sense for compound syncline. The broad shallow syncline of the Dakota sandstone falls in neither category. Objection might also be made to the use of

* Received Oct. 28, 1915.

† Published by permission of the Director, U. S. Geological Survey.

"isobaths" (lines through points of equal depth) to show altitudes. The argument in the upper half of p. 1344 does not seem clear and any possible interpretation of the words seems to involve fallacious ideas.

One of the first points made by Mr. Huntley is that the Dakota rests unconformably on various underlying formations, and the significance of this point is apparently brought out at the top of p. 1346 where it is stated that the "unconformity presumably eliminates the lower formations in that district as a source of such hydrocarbons." This point is of great interest because some investigators believe that unconformities play an important part in oil and gas accumulations by checking the rise of such fluids from below, though the fluids may be found both above and below the unconformity. To me it has seemed probable that oil and gas are much more frequently the product of downward concentration than is generally supposed and with regard to the Dakota in particular I am inclined to believe that Mr. Huntley is correct in his inference that oil has been carried downward by circulating water and has accumulated against the unconformity, the water passing on and the oil remaining behind in places where the pores or other openings in the rocks are large and the structure favorable. If this idea is valid, then in a sand of comparatively uniform porosity, which outcrops all about its border but is considerably higher along one side than the other, water should flow freely from the higher side of the border to the lower and might be expected to wash out most of the oil, gas, and salt water, so that the sand would contain and yield large amounts of fresh water compared with other sands. These inferences seem to be applicable to the Dakota sand and I have thought of it as one which might have been very productive of oil and gas had not circulation been so free and uniform throughout its extent, the reason for the free circulation being its uniform porosity, extensive outcrop, and the fact that one edge is considerably lower than the other.

The fact that the sand as a whole has a basin form does not seem to affect the results materially, because the basin is not so deep but that the hydraulic pressure along its higher or western edge is effective in moving the fluids throughout practically its entire extent. If the basin were deeper, or the pores on the whole smaller, the effect of the greater pressure along the western edge might be to cause most of the flow to take place about the north and south ends of the basin and if this had occurred the salt water, oil, and gas might not have been so completely washed out of the central parts of the basin. Again, if the porosity of the rock had been more variable it would seem that many more pools of oil and gas might have been retained in parts of the sand where the pores were larger, particularly where sand with such large pores occurred on anticlines.

If the source of the oil and gas had been below the Dakota sand the

results due to porosity, general eastward tilt, and extent of outcrop, might not have been very different, but under the hypothesis that oil and gas pools accumulate on anticlines from surrounding "gathering ground" without the aid of circulation, the results would have been of very different sort. The fact that far more carbonaceous material occurs above the sand than below, and that probably still more has been removed by erosion, and the fact that apparently no unreasonable assumptions are involved in the hypothesis that oil and gas have been carried downward into the sand, have inclined me to believe that such a process has been operative, and that had it not been for the free and extensive circulation of water through the Dakota sandstone, the rock might have contained many important pools of oil and gas. The fact is, however, as Mr. Huntley points out, that the sandstone is not altogether uniform in porosity but only comparatively so and that it has in places comparatively steep dips—spots which water could more easily flow around than across—hence there are places in the formation where oil and gas have accumulated, and may now be recovered.

At the bottom of p. 1335 the fresh water of the Dakota throughout large areas is attributed to lakes and musky swamps on a part of its western outcrop, "drainage conditions having kept its exposure more or less saturated with entering surface water since Cretaceous time." I think the Dakota would have contained fresh water if there had been no lakes or swamps on its outcrop and I know of no reason for supposing that the lakes and swamps referred to have existed for more than a very small part of the time that has elapsed since the Cretaceous period. The fresh water seems more reasonably attributable to the general permeability of the sand and its general dip eastward to an extended outcrop.

Although I am unable to follow and agree with all of the arguments in Mr. Huntley's paper, the conclusions stated in the summary, partly for reasons not set forth in the paper, seem in all probability correct. The fourth conclusion in particular is worthy of emphasis and elaboration. If, as I have long been inclined to believe, the source of most, if not all, of the oil of the region is above the Dakota, the oil once within the sand would tend to rise as high as possible in it and no matter what lateral movements took place later, there would be little chance of migration into lower sands where their beveled edges touch the Dakota. Another condition favoring the same result is the finer-grained character of the lower sands. On the other hand, if the prevalent view that the source of the oil is below the sand is correct, then pools should have been accumulated along the route of the upward journey of the oil in crevices and bodies of open sand surrounded with material having smaller pores. Once the oil is in such a sand as the Dakota, it would seem to matter little whether its tendency is to move up or down the dip; little or none should leave the sand for a lower stratum unless a considerable body of sand became completely filled with oil.

The Possible Occurrence of Oil and Gas Fields in Washington

Discussion of the paper of CHARLES E. WEAVER, presented at the San Francisco meeting, September, 1915, and printed in *Bulletin* No. 103, July, 1915, pp. 1419 to 1427.

MILNOR ROBERT, Seattle, Wash.—A year ago at this time, 12 wells were being drilled for oil in western Washington. Some excitement attended the operations; many thousand acres of land were leased to oil companies and stock was widely advertised. The location of the wells may be described by referring to the conclusions in Dr. Weaver's paper, in which the State is divided into four areas.

In the area described first, which includes the Cascade Mountains and eastern Washington, where conditions are decidedly against the existence of commercial oil fields, no new drilling was reported. In the second area, the Puget Sound Basin, where conditions are unfavorable, but to a less degree, five wells were being drilled. The third region, covering certain parts of southwestern Washington, contained four drilling wells and several others projected or beginning operations, in spite of the absence of petroleum seepages. None of the wells mentioned is fully active today.

On the western side of the Olympic Peninsula, along the Pacific Ocean, are three wells which have been the scene of operations for the past two to three years. The region they are prospecting contains seepages of light oils and is the fourth area described in the paper. The difficulties of transporting both men and supplies to the well sites have delayed operations and brought the costs up to high figures. The drilling of these three wells is interesting on account of the presence of oil in nearby seepages and its occasional appearances in the material penetrated.

A. C. McLAUGHLIN, San Francisco, Cal.—The most important part of Washington is on the Olympic Peninsula. The geological conditions on the Peninsula are not similar to the conditions in California, and it would not prove to be a continuous belt of oil-bearing territory from there to here if oil should be found at either end.

W. N. BEST, New York, N. Y.—What is the possibility of oil around Centralia?

MILNOR ROBERTS.—That is in the third district, where the Eocene and Miocene formations are present. The author regards that as neutral ground, without seepages, and with no indications of oil.

DAVID T. DAY, Washington, D. C.—Is the geologic structure there consistent with considerable accumulations of oil; and, secondly, have any bituminous shales been found in the Coast region of the State of Washington?

MILNOR ROBERTS.—I know of no bituminous shales in the whole region on the western side of the Olympics. The structure is considerably folded and somewhat faulted. It has variations in hardness and fineness of materials which might permit of cappings being present that would hold oil.

It would be impossible to say, in a general way, at what depth oil would be found, or at what depth capping would occur. The folds have put the cappings in such attitude that they can be studied very well and borings would go to varying depths, depending on the local topography. No ideal sites have been found for drilling, but anticlines are numerous. The covering of glacial material makes prospecting arduous and rather unsatisfactory, since it hides the bedding.

J. B. TYRRELL, Toronto, Ont. (communication to the Secretary*).—Professor Weaver has discussed the possibility of the discovery of oil in different parts of the State of Washington. The occurrence of vast pools of oil on the Pacific slope in the State of California has stirred the imagination of many of those living farther north, and during recent years some exploratory work has been done in the province of British Columbia in the hope of finding some northern continuation or representatives of the pools which have yielded so much oil to our fortunate neighbors on the south. In prosecuting exploration in British Columbia, some investigations have been conducted throughout the country near the mouth of the Fraser River, and some interesting results have been brought to light by these explorations.

In the southwestern corner of British Columbia, through which the Fraser River flows westward to the sea, there is a comparatively flat-lying area bounded on the north by granite mountains of Jurassic age, and on the east by the Sumas and Chilliwak Mountains, composed of somewhat older rocks. To the south this area extends across the International Boundary Line into the State of Washington, a rather high ridge of glacial débris extending east and west near the International Boundary Line. This comparatively small triangular area of about 700 square miles in Canada is underlain by Tertiary sandstones and shales with a total thickness of several thousand feet. These sandstones and shales do not appear to contain any fossil remains of marine animals, but they do contain trunks and fragments of trees and land plants which have been altered to lignite or low-grade coal, this coal occurring at numerous horizons throughout the formation.

The mode of formation of these Tertiary beds would appear to have been about as follows: In pre-Tertiary times a great submerged valley extended along to the south of the Jurassic Mountains. Into the head of this submerged valley or bay the early representatives of the Fraser

* Received Sept. 29, 1915.

River and other streams emptied, carrying down into it loads of detritus worn from the upper portions of the river valleys, and this detritus was spread over the floor of the bay, while around the margin of the bay grew forests whose trees fell into the edge of the water. While the streams were thus discharging their loads of sediment, the land was gradually sinking and the sediment was gradually accumulating to a greater and greater thickness. The trees growing on the shore fell in the water and were also included in this sediment, and finally formed irregular beds of lignite of greater or less extent, each bed marking the position of an old shore line of the time when its vegetation was buried. Thus the bay was gradually filled up with beds of sand and gravel, with included seams of coal or lignite, having a total thickness of several thousand feet.

After these Tertiary beds were deposited there was a rise of the land of 1,000 ft. or more, and the Fraser River cut out a canyon along the foot of the northern mountains to a depth of about 1,000 ft., and at the same time the surface of the adjoining land to the south was cut into hills and valleys, but there was no perceptible warping of the land in this district, and the beds retained the horizontal attitude in which they originally had been deposited. Subsequent to the formation of the canyon of the lower Fraser River, the land was again depressed to its present altitude, and the old canyon of the Fraser River was filled by glacial débris brought down from the mountains to the north of it.

At some time subsequent to the deposition of the Tertiary sands and clays, which are known as the Puget group, dikes of quartz-andesite were intruded through them near their northern border. These dikes also cut the beds of lignite, and subjected them to a process of distillation, giving rise to a dark oil which is still occasionally found exuding from the sandstone in the vicinity of the city of Vancouver and in the country to the east of it. It is not likely that this oil has any commercial significance or value, but its mode of occurrence as a distillate of lignite through the influence of hot dikes, is of interest.

The Occurrences of Petroleum in Eastern Mexico as Contrasted with Those in Texas and Louisiana

Discussion of the paper of E. T. DUMBLE, presented at the San Francisco meeting, September, 1915, and printed in *Bulletin* No. 104, August, 1915, pp. 1623 x 1638.

E. L. DEGOLYER, Norman, Okla. (communication to the Secretary*).—Referring to Mr. Dumble's description of the stratigraphy of eastern Mexico, I should like to question the classification of the Papagallos or Mendez shales as being of Cretaceous age and the so-called overlapping of the Papagallos or Mendez shales by the Alazan shales; the classification of the San Felipe series as Cretaceous, and the correlation of the San Felipe series with the San Juan limestones.

Mr. Dumble apparently recognizes the continuity of the great shale formation by his use of the terms "Papagallos" and "Mendez," as equivalent and interchangeable, since the Papagallos Hills in the State of Nuevo Leon and Mendez Station in Northern Vera Cruz, localities at which the shale formation outcrops, and from which the formation names are taken, are approximately 250 miles apart and on opposite sides of the Tamaulipas barrier. The Papagallos shales (evidently referring here to the northern extension only) are said to be non-fossiliferous and no reason for their Cretaceous classification is given, though in a previous article¹ they are classified as Cretaceous because at their northernmost outcrop they "have the same relation toward the overlying Tertiary that the Escondido (Cretaceous) has through its whole extent."

The Alazan fossil locality was found by me and the first collections from it were sent to Dr. Dall; about 17 species of marine invertebrates were recognized by him, the fauna noted as having a distinct Pacific tincture, and the age of the formation from which they came pronounced lower Eocene. The outcrop of the shale formation has been traced continuously from this point by me and my assistants into the Panuco River region where Eocene fossils are reported at a depth of 1,810 ft. in a well at Topila.² From Topila the formation has been traced continuously north to the Padilla-Jimenez region in Central Tamaulipas. Lithologically, the formation is fairly constant and though I am only generally acquainted with it north of the Padilla-Jimenez region, until definite evidence of the Cretaceous age of the Papagallos shale in its type localities, or of an overlap, is available, it seems to me more logical to consider the shale formation a single stratigraphic unit of Eocene age.

* Received Oct. 18, 1915.

¹ *Science*, New Series, vol. xxxiii, No. 841, pp. 232 to 234 (Feb. 10, 1911).

² E. T. Dumble: *Science*, New Series, vol. xxxv, No. 910, pp. 906 to 908 (July 7, 1912).

Such fossil evidence as is known to me regarding the age of the San Felipe series is meager and not very exact. Fossils from a depth of 3,150 ft. in Tamijuin well No. 3 were examined by Messrs. Hopkins and Belt and were said to have a decided Tertiary aspect. These fossils came from the San Felipe series as nearly as can be determined from the well log. Fossils from a depth of 2,900 ft. in Ganahl well No. 1, at the junction of the Moctezuma and Tamuin Rivers, were pronounced to be of Tertiary age by Dr. Hart. There is little doubt but that these fossils were from the San Felipe. On these classifications, the stratigraphic position of the formation, and lack of contrary evidence, I should classify the San Felipe as being of lower Eocene age. In addition to this evidence there are certain lithologic and stratigraphic features which have a bearing on such classification. The San Felipe series lies between the Cretaceous limestones and the known Eocene shales in northern Vera Cruz. It resembles a transitional series, being composed of alternating impure limestones and shales, often sandy, the limestones predominating near the base of the formation and the shales increasing toward the top until the formation grades so gradually into the overlying Eocene shales that it is almost impossible to determine the exact contact between the two formations where they outcrop and, in my opinion, quite impossible to determine the contact accurately from a well log; but it is apparently separated from the underlying Cretaceous limestone by an unconformity. Basal conglomerates, containing pebbles of Cretaceous limestone, in the San Felipe have been reported from several points in southern Tamaulipas and northern Vera Cruz and there is some evidence of the removal of the upper part of the Cretaceous limestones before the deposition of the San Felipe series in certain localities.

The San Felipe series apparently overlaps the San Juan limestones in southern Tamaulipas and its equivalent in north-central Vera Cruz, the San Juan extending perhaps into the Panuco-Topila region. The San Juan is absent from the excellent Boca del Abra section between Valles and Las Palmas, San Luis Potosi, and in the Potrero del Llano—Juan Cassiana region; the San Felipe series rests directly upon the massive bedded Tamasopo limestone.

Metallurgical Practice in the Witwatersrand District, South Africa

Discussion of the paper of F. L. BOSQUI, presented at the San Francisco meeting, September, 1915, and printed in *Bulletin* No. 101, May, 1915, pp. 997 to 1033.

SIDNEY J. JENNINGS, New York, N. Y.—I notice Mr. Bosqui says that zinc dust can be used profitably. I had occasion recently to try to ship some zinc dust to South Africa, owing to the high price of zinc in the last six or eight months and the difficulty in getting zinc sheets. We were informed that the steamship companies in England refused to accept zinc dust for shipment through the tropics on the score that it was an explosive mixture.

C. W. MERRILL, San Francisco, Cal.—In the early days zinc dust was packed in wooden casks. The steamers at that time were very chary of transporting it. It was put on the deck in the wooden casks, and if the zinc dust got wet it exploded. Since that time we have used hermetically sealed cases, in which case there is no risk whatever in its transportation. This rule of the steamship companies is perhaps another sample of their ultra-conservativeness.

HENNEN JENNINGS, Washington, D. C.—Unfortunately, I have not yet read Mr. Bosqui's paper, but believing it might interest you to know something of the beginning of the cyanide process on the Rand, I will speak briefly as to my own knowledge and experience of the early initiation of this process in South Africa.

I took the position of consulting engineer of H. Eckstein, the parent firm of H. Eckstein & Co., and a branch of the firm of Wernher, Beit & Co., at Johannesburg, in December, 1889, and at that time the Robinson Co. managed by them was the premier company of the fields. Up to this date plate amalgamation and catchment of a small percentage of concentrates by blankets, and in a few instances Frue vanners, were the main gold-saving devices used and had obtained an extraction of only about 65 per cent. of the gold contents of the ore. Also at that time the orebodies were seen to be changing from oxidized deposits to pyritic at varying depths from 50 ft. to 300 ft. below the surface, and uneasiness was felt as to future recovery. The deepest exploitation at any of the mines was not much below 300 ft.

Various treatments of the concentrates had been attempted by the Newbury, Vaughton, Pollock & Hungarian process, but with unsatisfactory results. Naturally I turned for further help to the best that had been done in this country in connection with the metallurgy of gold and which, at that time, was the careful concentration by vanners and treatment of concentrates by the Plattner chlorination process. Mr. Butters, at his works at Kenneth, had thoroughly established his ability and repu-

tation in this connection and he was selected to take charge of the erection of a chlorination plant at the Robinson mine. Mr. Butters arrived about the end of 1890 and started at once on the erection of a thoroughly up-to-date chlorination plant.

In the meantime the MacArthur-Forrest Cyanide Process syndicate representatives had given public demonstrations of their process at the Salisbury Co.'s mill in June, 1890. I was privileged to make the first test of this process. Mr. MacArthur and Mr. Forrest were employed by a firm of Scotch capitalists to go through the whole literature of gold extraction and if possible to obtain a practical and efficient commercial method of gold extraction. The result was the MacArthur-Forrest process, one of the few instances of a process made to order. Mr. MacArthur in no way attempted to disguise the fact that the dissolving of gold by cyanide solution and its precipitation on zinc plates was known to chemical students. His claim was that he demonstrated the practicability of using very dilute solutions and their precipitation not on acres of zinc plates but on compact mattresses of zinc shavings. Mr. MacArthur was also fully aware at that time of the factor of fine grinding in obtaining high extraction.

The South African rights of the process were in the hands of the Gold Recovery Syndicate of which Cordner James was head. Mr. MacArthur and Alfred James came to South Africa to demonstrate and introduce it, and they were present at the first tests made at the Salisbury Co.'s plant. The demonstration plant was constructed on the agitation principle and in the test that I followed through, both concentrates and tailings from the Robinson mine were treated. I remained two days and a night at the works. We obtained actual gold buttons which showed about all that was claimed for the extraction and, in fact, the extractions were in accord with the best subsequent practice. No information regarding costs was given, although I afterward learned it was between 20 and 30s. per ton.

As no information of costs was given, the Robinson Co. was unwilling to construct the first plant, but gave a liberal contract on a sliding scale for the syndicate to work about 10,000 tons of accumulated tailings, the Robinson Co. reserving the right to take over the plant at cost price and continue the working on a royalty basis, with a clause that no better terms were ever to be given other workers. The syndicate constructed the first working plant at the Robinson mines. At this plant, designed by Alfred James, agitators were given up and a system installed of working with square tanks and gravity percolation which contained all the vital principles of the great modern plants that have been improved and developed in details by numerous and various workers.

The syndicate put in charge of the Robinson plant George Darling, who was retained for many years as manager of the Robinson cyanide

works. At first the cost was as high as 13s. per ton, but it has gradually been reduced to below 2s. In the meantime Mr. Butters had started the chlorination plant at the Robinson and had done it in such an excellent way and made such a success of the works that he was requested by the Robinson Co., of which I was consulting engineer, to design and erect a new and additional plant for cyaniding at the Robinson, in conjunction with the old one. In this plant he made use of round tanks with bottom discharges, instead of square. The diameter of the tanks, due to my caution, was kept to the maximum of metallurgical practice, which, at that time, was only 23 ft. Large cement cyanide tanks, 40 ft. diameter, were first made use of at the Langlaagte Estate & Gold Mining Co., Ltd. and were designed by J. R. Williams who afterward became the cyanide manager of the Crown Reef Gold Mining Co. of which I was also consulting engineer; at a later date he there developed the first successful treatment of slimes on a large scale by the decantation method.

In this field Mr. Butters and Mr. von Gernet were also effective workers.

The Nigel Co., of which I was consulting engineer, was the first (in 1892) to deal successfully with the problem of pyritic tailings. The success of this plant was largely due to my assistant, Fearnside Irving. The direct filling of cyanide vats was first tried in 1893 under my direction at the Salisbury mill, which had square tanks, but the advisability and success of this method was only demonstrated in 1893 and 1894 at the New Heriot mine, with round tanks, at which W. K. Betty was cyanide manager and was working under my technical direction.

The method of filling the vats at the Salisbury and Heriot was by means of a hydraulic classifier and hose distribution in connection with slat discharges. Afterward a revolving circular distributing system was invented by Messrs. Butters and Mein and installed at the Robinson Co. Mr. Butters severed his connection with the Robinson Co. in 1892 and established a metallurgical company known as the Rand Central Ore Reduction Co. He did most excellent work in connection with this corporation and among other things gave a thorough trial to the Siemens-Halske electrical precipitation process. Mr. Betty's method of adding acetate of lead in the cyanide treatment made it hard for the Siemens-Halske process to compete with the zinc process.

It is needless to go further in the evolution of the cyanide process on the Rand, but without it few of the mines, which are now producing some 40 per cent. of the gold of the world and employing some 200,000 workers, would be in operation.

W. A. CALDECOTT, Johannesburg, South Africa.—The responsible position on the Rand occupied by the author during the past five years, his facilities for observation and experiment under working conditions,

the wealth of detailed ore-treatment data at his disposal, and his association with the introduction and trial of Merrill zinc-dust precipitation, Butters vacuum filters and other appliances, entitle his opinions to close scrutiny and careful consideration. The obligation conferred, however, upon his readers, would have been even further enhanced if the expression of his opinion in general terms had been more amply supplemented by definite figures upon such crucial points as percentage recovery by amalgamation, ore-treatment costs for more than one company, assay values of residues, etc.

The paper covers so wide a field that detailed criticism of all the points raised would occupy much space, and many features of interest, upon which various opinions exist, must be passed over. The author reaffirms his belief¹ in the general economic suitability of Rand methods to local conditions; possibly the "exhilarating atmosphere" elsewhere of numerous innovations is more congenial to the temperament of enthusiastic inventors than to the shareholders in the mines concerned. In other words, Rand practice has been developed in accordance with the axiom that "metallurgy is the art of making money out of ores," and the fiduciary position of those responsible for advising upon the expenditure of shareholders' money on reduction plants necessitates the relegation of other considerations to a secondary position.

The classification of variations in current ore treatment shown on p. 1000 is of interest, but does not include the disposal of current sand residue as a pulp down bore holes for mine filling, a system which is extensively and increasingly practiced;² nor is there any reference to the collection of slime, for both decantation and vacuum-filter treatment, being carried out in ordinary decantation vats; nor to the slime residue in either case being pumped as a fluid pulp to slime residue dams.

The proposed scheme (p. 1002) for dealing with the 9-mesh ore from the breaking station is an interesting variant from current arrangements, and virtually constitutes a separate circuit for classifying out the solids and returning to the trommels the water in this product. Some increase in capital cost and attendance will, however, be involved by its adoption, and the same object could be more simply attained by allowing the washings to enter a pulp launder in the tube-mill circuit, the coarser solids thus added being compensated for by running some stamps with finer screening or an additional tube mill during such time, and the additional water being dealt with by adequate safety cones following the tube-mill cones.

Doubtless, owing to the author's personal experience of Rand methods only dating from 1910, some confusion as to developments prior to that

¹ *Mining and Scientific Press*, vol. 108, No. 18, p. 736 (May 2, 1914).

² *Journal of the Chemical, Metallurgical and Mining Society of South Africa*, vol. xiv, p. 119 (Sept., 1913).

date is apparent in the suggestion on p. 1006 that the introduction of tube mills caused the development of heavy stamps.³ On the previous page the reason for the introduction of the latter is correctly stated as involving fewer crushing units with lower initial and operating costs, whereas the function of tube mills to secure cheaply any desired fineness of the crushed ore, and the coarse stamp-mill product most suitable for tube milling can be as readily obtained with a 1,250-lb. stamp as with a 2,000-lb. stamp. In passing, it can hardly be considered now (see p. 1013) that 9-mesh (0.272-in. aperture) is the economic limit of screen mesh, as 0.375-in. and 0.5-in. apertures are in regular and satisfactory use at the large Robinson Deep mill and up to an inch aperture has been successfully employed elsewhere.⁴ The reason why these last screens are not in more common use is that they permit a comparatively small increase in stamp duty above a $\frac{3}{8}$ -mesh aperture, and are liable to cause banking in existing launders of limited grade.

The statement on p. 1013 that in future at the Modder B a tube mill will deal with 264 tons per day of 9-mesh product mainly from ordinary stamps, whereas it can only handle 203 tons per day from Nissen stamps, indicates, assuming similar final pulp, that the screen pulp in the former case is much finer than in the latter, thus throwing less work upon the tube mill. Since the same battery screen is used in both cases, this illustrates the well-known fact that the stamp duty is no criterion of the work done in the mortar box, unless account is taken of the grading analysis of the screen pulp. The assumption on pp. 1006 and 1009 that 1,650 lb., and not 2,000 lb., is the economic limit of ordinary stamp weight is possibly based on trouble experienced through cam-shaft breakages when the latter were first employed with too high a number of drops per minute, but the fact that 1,900-lb. to 2,000-lb. stamps have been installed in nearly all recent mills (Consolidated Langlaagte, Government Mining Areas, Van Ryn Deep, Brakpan Mines, Roodepoort United, Main Reef and Modder Deep) shows that this difficulty has been overcome.

The flow sheet of a proposed mill on p. 1009 shows a double row

³ Compare remarks in Presidential Address by Prof. G. H. Stanley in *South African Journal of Science*, p. 46, November, 1914, as follows: "Tube mills were originally installed really to grind 'concentrates' from spitzluten to improve extraction; this so shortened time of treatment that separate treatment became unnecessary. Finer grinding was then adopted for all the ore; this led to coarsening the battery product, this in turn to abandonment of front plate amalgamation and the recognition of the stamp battery as a crushing rather than an amalgamating appliance, which at one time was considered to be par excellence. Next, in the effort to improve the output, the weight was increased, and percentage of water used decreased with consequent lowering of pulp elevating charges."

⁴ H. A. White, *Transactions of the Institution of Mining and Metallurgy*, vol. xxii, p. 185 (1912-1913).

of stamps, which is undesirable with heavy coarse-crushing stamps, both on account of the relatively limited bin capacity with high-stamp duties, and likewise because of the excessive lowering of the tube-mill cones required for the pulp from the back row of stamps. This loss of fall, involving additional pulp re-elevation cost, is accentuated by placing the cones of four tube mills low enough to receive pulp interchangeably, whereas three constitute an ample measure of interchangeability. As the diagram shows, the tube-mill cones are at a very low level, and the tube mills, plates, and return pumps consequently require a considerable depth of excavation.

In reference to the author's discussion of variants of amalgamation methods, the essential point appears to be missed, viz., that in modern mills the crucial factors of amalgamation recovery are the degree of liberation of the gold by the extent to which crushing is carried, and the efficiency of the pulp classification. The area of plates in the tube-mill circuit is quite secondary to these and mainly dependent upon the volume of pulp amalgamated, since with efficient classifiers gold particles of appreciable size merely circulate until amalgamated. The flow sheet of the proposed plant given in the paper shows that the overflow of the tube-mill safety cones joins the tube-mill outflow. With an imperfect arrangement of classifiers this may be necessary, and the cost of a considerable re-elevation of many thousand tons of overflow pulp daily in a large mill may be a warrantable expenditure in order to amalgamate gold particles overflowing the safety cones. The increased volume of pulp thus passing over amalgamated plates might require five of the latter per tube mill as advocated by the author, but when the tube-mill outflow only is amalgamated, three plates per tube mill have proven ample in practice under varying conditions on different mines, and at the Robinson Deep during 1914 with the design illustrated by Diagram C on p. 1018, the amalgamation recovery has amounted to 65.23 per cent. of the original gold contents of the ore, the sand residue averaging 0.315 dwt. per ton in value without ever showing any trace of free gold. It may be added that during 1914 this plant treated 576,370 tons of ore with a total residue of 0.252 dwt. per ton, equivalent to a recovery from all sources of 96.32 per cent., the slime original being 1.802 dwt. per ton and its residue after decantation treatment without air-lift agitation being 0.181 dwt. per ton. The appearance of free gold in the sand residue at the Modder B may be due to various causes, but so far as classification is concerned one small safety cone between two tube-mill cones is liable to considerable overloading with pulp if its feeding cones are overloaded, and the installation of return cones shown on p. 1022 for receiving tube-mill return pulp does not appear to be provided with safety cones, so that irregular distribution at this point is likewise liable to result in oversize particles or amalgam overflowing. On the other hand, the

simple classification system shown in C (p. 1018), whereby tube-mill and safety cones only are required and a minimum cost of pulp elevation is involved, permits of irregular or fluctuating distribution of pulp to the former to be corrected by safety cones of ample size receiving the total overflow of the tube-mill cones from a common launder.

In emphasizing the utility of Schmitt tube-mill feeders, the author expresses the common opinion of users but does not refer to the advantages of pebble bins with automatic pebble feeders actuated by the tube mill. He likewise confirms the advantages of the scoop discharge for tube mills as recently described in detail by W. R. Dowling⁵ and refers to the desirable location of tube mills below stamps as illustrated by the same metallurgist over four years ago⁶ in the form shown in Diagram C.

As the scheme for a proposed new reduction works illustrated in Fig. 7 is not in actual operation, comment thereon can only be based on past experience of the various devices shown and on their relation to each other; it is to be regretted that the author had not the opportunity of demonstrating by actual results the advantages or otherwise of his proposed plant as compared with existing practice.

The undesirability of a double row of heavy stamps and the excessive volume of pulp to be amalgamated and reelevated has already been referred to, as well as the absence of safety cones following the return cones. The elevated position of these last further involves much coarse pulp being raised to them merely to gravitate down to the tube-mill cones, instead of being elevated only to the height of the latter, according to the method shown in Fig. C on p. 1018.

The Dorr thickener shown, while a useful device when crushing with cyanide solution for removing surplus solution from slime pulp prior to air-lift agitation or vacuum filtration, is not adapted to effect as complete a separation of water from settled slime as the ordinary intermittent settlement and decantation. The author's assumption that the latter yields a settled slime containing 50 per cent. of moisture with blanket ore slime, which is a very good result for a continuous thickener, is apparently based on data from obsolete and inadequate plants, since in a modern slime plant 40 per cent. of moisture in settled slime is by no means the highest degree of settlement obtained under ordinary working conditions. This difference of moisture content is of great importance, because with a 50 per cent. settlement half as much water again is added to the working solution as with a 40 per cent. settlement, and in consequence the same additional amount of dilute working solution has to be regularly discharged with its gold, cyanide, and lime contents.

⁵ *Journal of Chemical Metallurgical and Mining Society of South Africa*, vol. xv, p. 214 (March, 1915).

⁶ *Journal of Chemical Metallurgical and Mining Society of South Africa*, vol. xi, p. 421 (March, 1911).

As regards the vacuum filter, the author places the extraction by this process at 93 per cent. This is appreciably higher than by decantation, but is obtained at an extra cost of $2\frac{1}{2}$ d. to 4d. per ton. Comparing this with the Robinson Deep figures quoted by the writer earlier, which show an extraction by decantation slime treatment of practically 90 per cent., it will be seen that the difference of 3 per cent., equal to 2.7 d. on 1.802 dwt. slime, is slightly less than the average extra cost of filtration. Hence the author's dictum that vacuum filtration should be adopted in all new plants may be received with some qualification, and as being specially applicable to rich slime where a small additional percentage recovery on the value is profitable.

The inference on p. 1024 that thorough separation of sand and slime by classification is due to the influence of vacuum filtration is hardly warranted by present or recent practice, since the achievement of such separation has been in vogue prior to 1910,⁷ on plants where decantation practice obtained and is still practiced. In fact, at the present time the most advanced practice is to confine the product separated as slime to colloid particles so far as possible, since fine granular particles of silica or pyrite yield a very good extraction when included in the sand charges without interfering with leaching, whereas their presence in slime charges delays dissolution of its gold contents, raises the value of slime solution, and increases wear on slime sludge pumps.

Fig. 7 does not indicate the disposal of the water drainage from the sand vat when filled, but the proposed treatment in this vat of the sand collected from a water-borne pulp is a revival with cleaner sand of Rand practice of 15 or 20 years ago.⁸ As the author states on p. 1025, it was formerly customary to apply weak cyanide solution in the sand collectors, but he omits to give the reason for the abandonment of this practice, which was the contamination of the mill service water with cyanide and consequent dissolving and loss of gold contained in the ore crushed. The seriousness of this loss may be gauged not only from the above fact, but from published references,⁹ of which possibly the most notable are the following, *Transactions of the Institution of Mining and Metallurgy*, vol. xiii, pp. 77 to 84, Nov. 19, 1903, by Charles Butters and Hennen Jennings. Charles Butters said:

No cyanide solution is ever run into the collecting tank (i.e., when equipped with a Blaisdell excavator) as its contents, after having been filled from the battery, are allowed to drain, and are then transferred by means of the excavator direct to the

⁷ *Journal of Chemical Metallurgical and Mining Society of South Africa*, vol. x, p. 51 (August, 1909).

⁸ F. C. Pengilly, *Transactions of the Institution of Mining and Metallurgy*, vol. vi, p. 113 (Jan. 19, 1898).

⁹ A. von Gernet, *Journal Chemical Metallurgical and Mining Society of South Africa*, vol. ii, p. 529 (January, 1899); and H. T. Pitt, vol. v, p. 84 (September, 1904).

leaching tank. Therefore no loss of gold can take place from cyanide solution having been introduced into the collecting tank, as is the case when the collecting tank is filled directly from the battery, and gets its preliminary treatment of cyanide solution in the same tank. I have yet failed to find a reservoir on the Rand that did not have gold in solution from this cause. This gold in the reservoir varies in value with the season—being higher at the end of a dry season than at the commencement of it, because during the dry season the reservoirs never overflow, but become concentrated from evaporation in the warm atmosphere.

At Virginia City we found as much as 50 per cent. of gold dissolved out of the charge, by filling into a collecting tank in which the charge before had received a preliminary cyanide treatment, and, strange to say, this loss continued for over two weeks after we had discontinued the use of cyanide in the filling of the vat, and this notwithstanding that we had washed it out as clean as we could. While we could obtain no trace of cyanide at any time, still the loss went on, until after two very anxious weeks it gradually stopped. This occurred on two different occasions. At first I attributed it to an excessive amount of ferric salts in solution, acting possibly as a solvent for gold; but we found out later that it was from the traces of cyanide that could not possibly be detected by any chemical test.

Mr. Hennen Jennings and myself had both made a series of determinations at Johannesburg as to the gold in the battery water from this source, and with similar results—gold was invariably found. It is not easy to state the exact loss from this source, but that it is a source of loss which should be eliminated I think all engineers and metallurgists will agree.

Hennen Jennings said:

The remark of Mr. Butters as to gold being found in the battery water was correct. He regretted that he had no detailed figures with him. Some very extensive tests had been made at mines with which he had been connected, and it had been found that even in large storage reservoirs the water held a slight amount of gold in solution. It varied from a trace up to sometimes as high as 1 dwt., but that was very exceptional.

That a good extraction from clean sand can be obtained by cyanide treatment of sand in the vat in which it is deposited from a solution-borne pulp is evident from the results with this method which is in present operation on the Rand at several mines.¹⁰ This fact, however, does not dispose of the point that actual recovery of the gold and not merely extraction from the ore is the true objective, and the author does not disclose how he proposes to overcome the contamination of mill service water circuit and the consequent loss of gold, resulting from alternately introducing into the same vat and saturating the filter cloth with water and cyanide solution. Roughly, the consumption of water in a Rand reduction plant is half a fluid ton per ton of ore milled, and this loss mainly occurs as moisture in sand and slime residues. In addition, however, water is consumed by mine-filling with sand residue pulp, by the periodical discharge of slime residue with mill service water, by lia-

¹⁰ *Transactions of the Institution of Mining and Metallurgy*, vol. xxii, p. 182 (1912-1913), (which shows a sand residue of 0.233 dwt. per ton by single treatment in the collecting vat).

bility to seepage and overflows at some point in its circuit, and by incidental use for many minor purposes; in general, long experience has emphasized the desirability of maintaining the mill service water as free from gold as possible, as stated by the authorities quoted above, if mysterious shortages of gold called for are to be avoided. The maintenance of free protective alkali in the form of calcium hydrate in mill service water on the Rand prevents the rapid decomposition of traces of cyanide by acid iron salts in the ore, and since sodium cyanide is capable of dissolving about twice its weight of gold, the presence of only 0.0001 per cent. of sodium cyanide in mill service water may be equivalent to over a pennyweight of dissolved gold per ton of water.

A. J. CLARK, Lead, S. D.—One point I think ought to be made is this: That the establishment of the decantation process creates a large volume of low-strength solution which is successfully precipitated by zinc shavings, where zinc dust has made only a qualified success. I think this point needs emphasis, that zinc shavings would not precipitate these solutions for any length of time. New shavings are used for a very few days and then the shavings are transferred to the box in which the original solutions of the sand plant are treated. The process in Africa thus conforms to the requirements of zinc-shavings precipitation, but the zinc-dust process has to stand by itself in each instance. The zinc dust cannot be used again. It is not possible to take it from the press and use it in a stronger solution successfully. At least I have never found it so.

C. W. MERRILL, San Francisco, Cal.—That is an interesting point. It shows how the process is developed according to local conditions. In Africa they have a mild climate and cheap labor, but up at the Homestake we have a very severe climate and expensive labor. We developed methods whereby we were able to handle larger bulks of pulp and still not exceed the cost factor. Local conditions will to a certain extent determine the trend of the process.

G. H. STANLEY, Johannesburg, South Africa (communication to the Secretary*).—It might be well to emphasize the great improvement which modern results in cyaniding show in comparison with early practice as deduced, for instance, from examination of old dumps.

In several cases the interiors of these have been laid open to examination as a result of sand-filling operations by hydraulicking, etc., and assays have shown the presence of thousands of tons assaying as much as 3 dw. per ton gold in the older and bottom portions, while even the newer portions, though still made prior to tube-milling days, give assays fre-

* Received Sept. 2, 1915.

quently approaching 1 dwt. and in some cases considerably in excess of that figure.

One dump of about 1,500,000 tons averaged 1.2 dwt., another of over 1,000,000 tons averaged 1.65 dwt., and these values have probably been reduced from the original by years of atmospheric leaching in the presence of thiocyanate in the acid dump. This high value was undoubtedly due in great measure to the coarseness of crushing which obtained, grading tests showing the presence of 35 to 45 per cent. + 60 sand particles, but at the same time a large proportion of slime is present, often in distinct layers, which prevented proper treatment of the sand and itself received practically no treatment on account of impermeability.

Slime washed from the sand and assayed showed values generally about twice as high as the average of the dump. The washing was consequently very imperfect as is evidenced by the frequent presence of patches of Prussian blue color.

These points are, of course, remedied in modern practice by fine grinding to practically all through 60 mesh and by better classification, so that at present a good sand residue may assay only 0.25 dwt. Of course, the original value of the sand treated was formerly much higher than obtains at present and the extraction percentage was apparently fairly satisfactory, but that the high residue value was not due simply to the high original value is shown by the fact that a residue value which would be considered extremely good in current practice can be obtained by regrinding and retreatment of this old residue. The retreatment of these old residues is, however, in most cases precluded by their being covered up by later low-grade residues; by their great acidity, necessitating the use of excessive amounts of lime; and other causes.

Sand dumps are regarded as a necessary evil, and give rise to much discomfort by the dissemination of dust in windy weather. Various means have been tried for preventing this, the most successful depending upon binding the particles together by spraying the outer surface of the dump with black veil mud or ore slime and salt mixed with water, both these methods being now in effective use on one or two favored properties.

There was obviously in the early days every incentive to treat with the sand as large a proportion of slime as possible, otherwise it received no treatment at all. Consequently the introduction of the decantation process by Mr. Williams constituted a great advance indeed. Before making my first personal acquaintance with the method (in 1905) I was well aware of the theoretical imperfections of the method and had busied myself with the development of a suction filter. It did not require a very long while of actual contact though to convince me that my suction-filter idea (on much the same lines as the present Moore or Butters but at that time unsupported by experience) had little chance of adoption in place of the very cheap and effective operation of the decantation plants.

The margin for improvement in saving was apparently so small; and, although in the few years following much experience was gained elsewhere, only a very confident technical adviser and very strong financial interests could afford to take the respective risks involved. I am very glad that the introduction of suction filters has been so well justified.

Nevertheless there seems to be no prospect of the whole of the ore being treated as one product; local feeling can be summed up in the statement that while we know slime cannot be treated like sand, we hold that sand is best treated as sand and not as slime.

As with suction filtration, so with precipitation, and I am in complete agreement with Mr. Bosqui in his opinion that this is the department in which the most notable advances can and will be made. The present method has always appeared to me to be the weakest part of the whole sequence of operations and I am confident that if the same amount and quality of attention had been devoted to electrical precipitation as to zinc precipitation, a better method would have been evolved. Of the two, the zinc method appears more understandable to the practical worker deficient in scientific training and the variable factors in its operation are more or less within his ken, so that he can easily experimentally vary the conditions and ascertain those most suitable, while in the electrolytic method expert scientific knowledge and appreciation of the effects of varying factors are much more urgently required.

Experimental work in this direction has been carried out by graduate students in my laboratory and I understand that their work is being extended on one of the mines. I am certainly hopeful of the results.

Breaking and Sorting.—Sorting and breaking are commonly done at the shaft head in the "crusher station," and the broken ore is then transported to the mill, either in hopper-bottom cars, often of full railroad size, drawn by steam or electric locomotives running over the battery bins, or alternatively conveying and elevating belts are used.

The non-crystallization of design into some standard method is partly explained by the difference in the amounts of waste sorted out at different mines, which in turn depends considerably upon the variation in thickness and dip of the reefs. This variation affects the relative amounts of rock broken by machines or hammer boys and hence the amount of waste mined.

Undoubtedly the use of circular sorting tables in buildings with several superposed floors gives rise to crowding in a breaker station; while with belts the building is spread out over much more ground space, there is more room available, and better lighting is possible, which is in favor of the employment of belts.

I must confess to a feeling of preference for jaw breakers as opposed to gyratory crushers, mainly on the ground of greater ease in clearing after a choke or sudden stoppage, and of renewal, replacement, or repair.

Mr. Bosqui does not give the size to which the ore is broken for stamp milling, and although about 2 in. is aimed at, the feed in most cases is actually nearer 4 in. Jaw breakers for this size are satisfactory and the larger size is apparently of no moment with modern heavy stamps.

Stamp Milling.—Here the Rand has little to reproach itself with on the score of conservatism. The figures given relative to the superior performance of Nissen stamps appear to be conclusive.

Amalgamation.—Mr. Bosqui appears to cavil at the extreme to which reduction in the number of tube-mill plates has been carried on the Gold-fields group. This of course is accompanied by efficient classification so that such portion of the pulp as is bypassed to the cyanide plant will only carry a little free gold and that in a finely divided state; so that in the first place it would only be amalgamated with difficulty on account of its slow settlement and in the second it is easily soluble in cyanide.

It is commonly stated that gold amalgamated represents 100 per cent. extraction, while if it gets to the cyanide plant only 90 per cent. can be recovered, and this is one of the half-truths so difficult to controvert. The residue from cyaniding contains gold partly undissolved (because incased, etc.) and partly as dissolved gold which has not been washed out, the former being much the more important in amount. The finely divided free gold referred to, which presumably might have been amalgamated, will presumably also be completely dissolved in cyanide and nearly all, not 90 per cent. only, removed and recovered; the value in the residue is mainly due, as stated, to incased gold which could not have been amalgamated either. There is, of course, no need to point this out to the author, but it may not be obvious to some of his readers.

Practically all the free gold which would be able to settle in the stream of pulp passing over the plates and so become amalgamated, settles in the classifier and therefore gets on to the plate, as is shown by the very small difference in recovery when bypassing and using few plates, or when using the alternative practice, in each case under good conditions.

In connection with the trials at Modder B, we are not told whether any pulp was bypassed and I suggest that the appearance of free gold in the residue indicated merely overcrowding of the plates with pulp and that a cure might have been found in bypassing some, or more, of the pulp—the carefully classified finer or lighter portion, of course—and should like to know whether this was tried. This view obtains support, I think, from the fact that overcrowding of the full 30 plates also led to the appearance of fine gold in the residue.

Tube Milling.—Mr. Bosqui on p. 20 refers to the broken quartz necessary to maintain the pebble load. I merely mention this as it might give a wrong impression—it should, of course, read, ore or banket.

With regard to peripheral discharge, similar experiments to those instanced at the City Deep have been in progress elsewhere and the results are given by Mr. Dowling.¹ As a result the wide adoption of scoop discharges seems to be foreshadowed, but will necessitate the provision of increased power. Mr. White has also put on record before the same Society results of experimental work with a short section of a tube mill of variable diameter, which I had the pleasure of superintending.

In the case of the tube mill with ball mill end described, it would seem that the fact that partitioning off the end of the mill in that manner entails no decrease in efficiency constitutes an argument in favor of the view that the length of mills might be lessened. The few balls would have little action on the already fine pulp, certainly much less than the usual charge of pebbles; their action would be practically confined to the small pebbles to be crushed.

Classification.—The author's conception of the mode of action of a diaphragm classifying cone appears to be quite exceptional and I hesitate to accept it. I should like to ask whether he has any evidence that the outer portion of the settled sand is crowded upward? The central portion is certainly coarser and therefore heavier so far as the weight of individual particles is concerned; but I have often noticed that the outer and finer portion appears to be much richer in pyrite and consequently the specific gravity may be even higher than in the center. Unfortunately, I have no figures available. My impression of the mode of action is that the concave shape of the top surface is determined by the fact that the disturbance at the central feed is too great to allow any but the very coarsest pieces to settle; as the pulp flows out toward the periphery the velocity decreases so that more and more sand of successively finer grade settles out, the feed being so regulated that only particles sufficiently fine reach and are carried over the edge, but that all the settled sand is slowly settling toward the outlet. The coarse center tends to move faster than the sides and this would lead to "break-aways" if it were not checked by the diaphragm which serves both to take pressure off the outlet and to mix the sand in passing by its edge.

The separation of sand and slime is more difficult than the preparation of tube-mill feed, as the particles are more nearly equal in size. Where the same kind of cones are used, the underflow (sand) is considerably contaminated with slime, and is diluted with wash-water and reclassified, using in each case a thick underflow; or else, as the author describes, a separate clean water service is fed to the bottom of the cone, inside, which prevents slime issuing with the underflow sand—

¹ *Journal of the Chemical, Metallurgical and Mining Society of South Africa*, vol. xi, p. 414 (March, 1911).

washes it, in fact—but then uses a thin discharge; in either case the final sand contains very little slime, and it therefore becomes possible to add the required lime in the battery as the difficulty of premature slime settlement in sand is thus obviated.

Treatment of Sand.—There must be few plants operating as described under method 1. It is usual, where hose filling is used, to pass the mill pulp through spitzkasten or spitzluten, and convey the spitz underflow, still containing much slime, to the collectors through hose. It might be mentioned, too, that at some plants a considerable proportion of the residue is disposed of by sending it immediately underground for stope filling.

In connection with the filter-table method of sand collection, the sand is ploughed off into cyanide solution and then pumped to collecting vats via Butters' distributors; it is consequently agitated and collected in cyanide solution. Dissolution of the gold is thereby facilitated, and treatment time decreased, necessitating less plant and consequently economizing capital. In this connection it may be mentioned that for a given daily capacity the cost of a reduction plant is now only about one-third of that before the war.

Slime Treatment.—I have already referred to the decantation method and would only add now that the chief trouble met with is unsatisfactory settlement, particularly in cold weather, and on one or two plants this has been overcome by Mr. Salkinson's method of warming the pulp by the utilization of waste steam from the hoisting or mill engine, this incidentally increasing the capacity by as much as 20 per cent. Unfortunately the general use of purchased electric power makes this impossible on new plants.

With regard to air agitation it should be pointed out that the use of the usual tall, narrow Brown tanks is not universal. Alternatives are the employment of tanks about 30 by 30 ft., in which the central air lift does not extend to the top, and which it is claimed use therefore less power; and in some very large vats there are several comparatively narrow air-lift tubes, with tangential discharge, so as to set up a swirling motion of the whole pulp in the vat, and prevent too rapid settlement.

In connection with the Usher process it was understood that the vat acted as a huge classifier, so that the charge was not homogeneous and correct sampling became almost impossible.

Smelting.—From either method the slags still carry several ounces of gold per ton, even after crushing and panning, and even on plants where "pot" smelting is employed a pan furnace is often installed for smelting these slags with lead, in order to recover this gold; at the same time any other gold-bearing byproduct about the plant may be similarly dealt with for the extraction of its gold.

Even after this, the slags obtained may receive a further treatment

by smelting in a blast furnace, two or three plants having their own, but more usually selling it to a byproduct works.

There were formerly several of such works, operated by independent people, but a coöperative byproduct treatment works has also been established by the gold-mining companies, to which those companies which do not possess their own facilities dispose of the byproducts, which are there treated as at the larger mines.

In conclusion, while there is uniformity in the main outlines of practice, it will be evident that many variations in detail exist among the 50 producing mines.

This is not due entirely to the idiosyncrasies of controlling officials, for the ore itself, and other circumstances, vary somewhat from mine to mine, apart from variations in value. In some places it is harder than in others; again, the nature of the country rock exercises an influence, sericitic schist, for example, adversely affecting amalgamation, and in some cases a small proportion of other metallic minerals occurs in the ore, and doubtless has some influence; I have, for instance, found traces of telluride in some cases.

It will be seen that no concentration, either "clean" by tables for subsequent chlorination, or simply rough for separating part of the pulp for longer cyanide treatment, is now employed, and, in consequence, chlorination has been completely discontinued.

There is, however, a small proportion of very clean and rich pyrite collected chiefly with amalgam, after separation from amalgam and grinding finely in small tube mills, etc., which is treated simply by agitating by air in a special small cyanide agitating plant—as first suggested by A. F. Crosse—and a large percentage of the gold is thus extracted and dealt with as usual, the residue commonly going to the main cyanide plant.

This pyrite contains appreciable amounts of osmiridium, which may possibly be turned to account in the future.

W. R. DOWLING, Germiston, South Africa (communication to the Secretary*).—Under "Classification" on p. 1017 the author is somewhat in error in describing the earlier classification methods. The inverted pyramidal spitzkasten were used merely to remove some of the water in the tailing pulp, although incidentally some slime was also carried over, and the underflow being reduced in volume was more easily handled in the sand collector vats where the main separation of sand from slime took place.

On pp. 1019 and 1023 the author refers to the use of the vacuum sand filter table as being a method for obtaining clean sand. Obviously

* Received Oct. 7, 1915.

this cannot be the case, as the filter table merely dewatered the clean sand previously obtained by cone classification. The sand being thus freed from water is then ready for cyanide treatment, without fear of complications arising through the introduction of cyanide solution into the mill water. The author when discussing the separation of sand from slime somewhat confuses the question by introducing the classification of pulp for tube milling into the same paragraph. When analyzed, the statement in question appears to mean that the separation of sand from slime was to be effected in large cones in the same way as with those already in use with sand filter tables, except for the use of water at the apex of the cones in the former case. For its proper treatment, sand should in any case be free from slime, to the extent at least that layers and lumps do not exist, but the author is in error in holding that "an imperfect knowledge of classification, or a failure to recognize its importance, made this scheme (i.e., the treatment of sand in the collecting vat) impracticable." That the classification in the light of later knowledge was imperfect is true, but an equally important cause for the abandonment of sand treatment in the same vat as collected was the fact that gold in solution was lost. Quite apart from any accidents due to solution being pumped on to a vat with leaching pipe or overflow doors open to water storage vats, or similar accidents by which gold-bearing cyanide solution entered the mill water, there were the difficulties of removing the gold-bearing cyanide moisture from the inch or so of sand usually left in the filter bottom when a vat is emptied of sand. The cocoanut matting and jute forming the filtering medium were similarly saturated, while the bottoms of steel vats are irregular and a series of solution pools formed. It was only when regular and systematic samplings and assays of mill water were made that these losses of gold were appreciated. I well remember my attempts, in 1899, to remove gold-bearing cyanide solution from the filter bottoms of collector vats by repeated small water washes. Although the washing reduced the loss appreciably, the extra labor and expense were considerable. It was impossible to absorb these washings into the plant as the moisture of discharged residue was the same as collected sand, so the liquid after precipitation was run to waste with consequent loss of water. This solution, by the way, was rendered acid before precipitation on iron or zinc, and the gold recovered in a precipitation plant set aside for the purpose. The mill service water, on the plant referred to, as in most earlier and some existing plants, was stored in earthen dams, and the loss by seepage of gold-bearing water must have been considerable. Although various precautions may be taken against the loss of mill water, such as the use only of vats for storage, and against the introduction of gold-bearing cyanide solution into the circuit water, these only minimize the evil and the fact still remains that the milling will be done in a dilute cyanide solution. The question then naturally

arises as to what is the real objection to milling in cyanide solution. I have searched in vain for a pronouncement by the author on this matter, where he is treating an ore yielding so high a proportion of its gold contents by amalgamation and which gold, as he rightly points out on p. 1010, is only commercially recoverable on plates.

The real objection to milling in cyanide solution on the Witwatersrand is that since the ore yields so high a recovery by amalgamation, indicating relative coarseness, the treatment of the ore by cyanide only would naturally be prolonged, expensive and imperfect. Should amalgamation be carried out nevertheless, complications due to corrosion of copper plates and dissolution of mercury arise. Both of these metals are subsequently precipitated on the zinc in the extractor boxes and result in base bullion, due to copper. Furthermore, crushing in cyanide solution renders it impossible to obtain accurate screen samples by reason of the presence of dissolved gold; and the cyanide goes on dissolving gold from the sample itself. It is an interesting fact that although a few mines did construct plants to mill in cyanide solution, one of the plants reverted to water, and no responsible engineer or metallurgist has since been bold enough to advise his principals to adopt this method.

Notes on Homestake Metallurgy

Discussion of the paper of ALLAN J. CLARK presented at the San Francisco meeting, September 1915, and printed in *Bulletin* No. 103, July, 1905, pp. 1381 to 1398.

G. H. CLEVINGER, Palo Alto, Cal.—Since the precipitation of extremely dilute solutions as regards cyanide presents one of the greatest problems of zinc-dust precipitation, I should like to ask Mr. Clark the minimum concentration in cyanide of the solution from which it is possible to precipitate satisfactorily the gold with zinc dust in Homestake practice.

A. J. CLARK.—For a day's work about 0.015 per cent. That is precipitating a very low-grade solution, very low in alkalinity. It requires probably 0.2 lb. of zinc dust to the ton.

JOHN A. FULTON, Melones, Cal.—At the Melones mine, we have been working very low-grade rock for a number of years, the process being amalgamation and concentration up to the last year. The value of the ore increased so that we had to supplement the concentration with a sand and slime plant. Classification was a problem, since to install classifiers would mean tearing the mill all to pieces. We are therefore making use of a Wilfley table for this purpose. The amount of wash water is increased and a stream of fresh water is also run into the feed box at the

end away from the intake; this gives a clear line of demarcation between the sand and the slime, enabling us to draw off sand at one end of the table and slime at the other.

A Rule Governing Cupellation Losses

Discussion of the paper of W. J. SHARWOOD, presented at the San Francisco meeting, September, 1915, and printed in *Bulletin* No. 104, August, 1915, pp. 1671 to 1673

FREDERIC P. DEWEY, Washington, D. C. (communication to the Secretary*).—Even if the data given should be considered sufficient to support properly the empirical rule as stated, this rule would be of extremely limited application in miscellaneous assaying. The fixed conditions must include equality of gold and silver where mixed beads are concerned, owing to the wide variation in the specific gravity of gold and silver. Also, the fixed conditions must include the presence of the same amounts of other base metals besides lead. Such an agreement is often lacking even in the beads from two fusions of the same ore. In practical work it is sufficiently general to be called universal that cupellation beads are not pure precious metal. An apparent loss in cupellation is an algebraic sum and in many cupellations the gains exceed the losses, especially with large beads where the gains lie so completely in the judgment of the cupeller.

For several years I have been engaged in an investigation into the conditions affecting the accuracy of gold-bullion assaying, and have just completed an examination of cupels which included the assaying of 10,000 used cupels. The grand result of the cupel work is an emphatic warning against drawing rigid conclusions regarding cupellation. It is so easy to say "If all other conditions remain the same," but it is extremely difficult, and in practical work impossible, to maintain equal conditions. Cupel absorptions, which constitute such a large proportion of cupellation losses, and in many cases the only loss, may vary considerably in different parts of the same row of cupels, even in carefully regulated work.

* Received Sept. 1, 1915.

Recovery of Mercury from Amalgamation Tailing, Buffalo Mines, Cobalt

Discussion of the paper of E. B. THORNHILL, presented at the San Francisco meeting, September, 1915, and printed in *Bulletin* No. 104, August, 1915, pp. 1653 to 1670.

D. B. HUNTLEY, Oakland, Cal.—About 15 years ago it chanced to be my lot to cyanide some mill tailings, assaying about \$5 per ton in gold and a few cents in silver. It was in southern Idaho, a desert region where costs are nearly as large as in central Nevada. We knew that we had a lot of tailings that had been amalgamated and contained about 1 lb. of quicksilver per ton. We knew that we would dissolve some of the quicksilver when we cyanided them with say 0.25 per cent. cyanide solution. We figured (optimistically) that the cyanide would dissolve all of the quicksilver, and we thought to capture all of that, 1 lb. per ton; and we figured out profits as partly gold, partly silver, and partly quicksilver from the recovery in working those tailings. Actually, however, we did this: We had a battery of three large, old-fashioned silver-mill retorts. We retorted our zinc-dust precipitate, recovering about $\frac{1}{4}$ lb. of quicksilver per ton treated. That was only a loss of 75 per cent. of what we originally had, but still it was profitable by virtue of the conditions. I mention it partly to call attention to the crude old way of doing it, and partly to call attention to how chemical methods for recovering quicksilver have developed in the last 15 years.

H. G. S. ANDERSON, San Francisco, Cal.—It may be of interest to mention, in reference to what the previous speaker has said concerning his experience in southern Idaho, that we tried for a while to precipitate silver from solution with zinc shavings, but we had five compartments in each box; after 30 min. there would be no precipitation in the first compartment at all; the mercury, precipitating before the silver, would cover the shavings in the second compartment in about $\frac{3}{4}$ hr., and in from 3 to 4 hr. there would be no precipitation taking place in the entire precipitation box. Then we turned to zinc dust, and zinc dust did not give us as clean a silver bullion as aluminum did, so finally, although the cost was a little higher, aluminum was accepted as the precipitant and the bullion was in much better condition and easier to smelt than it was either in the case of zinc shavings or of zinc dust.

Electric Furnace for Gold Refining at the Alaska-Treadwell Cyanide Plant

Discussion of the paper of W. P. LASS, presented at the San Francisco meeting, September, 1915, and printed in *Bulletin* No. 103, July, 1915, pp. 1443 to 1447.

ROBERT M. KEENEY, Somersville, Conn. (communication to the Secretary*).—There is not only a possibility of improvement in present methods of treatment of the byproducts of refining in the cyanide plant by the use of the electric furnace, but also in melting and refining of gold precipitate. This interesting paper by Mr. Lass shows an electric furnace in successful operation in a large plant for the treatment of byproducts. Electric melting of cyanide precipitate is a cheaper form of treatment in many cases than any of the combustion methods now used because of the high cost of oil, coke or coal in many camps, while on the other hand electric power may be had cheaply, especially in a place where there is possibility of development of high-head hydro-electric plants for power purposes. Also, if the oil-fired tilting furnace is used for direct melting of the precipitate, as is the case at many small mills, a great saving in cost is made by the use of the electric furnace because of the high crucible cost in melting with the-tilting furnace.

A study of this paper indicates the cost of the operation approximately. Consider all the fluxes to be turned in as byproducts and coke at \$18 per ton.

Cost of Electric Smelting of Byproducts at the Alaska-Treadwell

	Cost per Ton
1,044 kw.-hr. at 1c	\$10.44
Labor.....	5.25
20.4 lb. graphite electrodes at 18c	3.77
404 lb. coke.....	0.36
Total.....	\$19.82

It would have been interesting to have had figures on the gold losses. Unless there is a higher extraction in the electric furnace which would offset the cost of smelting, it is probable that in this case blast-furnace smelting of the byproduct material is the cheaper method.

Recently I made an investigation of the cost of melting precipitate from the zinc-dust presses in the electric furnace as compared with the cost of the present process used at a certain mill, melting in an oil-fired tilting furnace. The precipitate is now dried in a wood-fired muffle furnace, then mixed with the proper fluxes and melted to bullion. At

* Received Aug. 24, 1915.

periods the ore entering the mill carries considerable copper which enters the solution and comes down in the precipitate. The bullion is from 200 to 600 fine in gold and silver, of which about half is silver. The mill is 25 miles from the railroad so that the cost of fuel oil is high, 25c. per gallon, and the cost of coal and coke so high as to prohibit their use. An abundance of wood is available from which charcoal could be made, but the cost of this would be close to \$20 per ton. Hydro-electric power was developed in the district at a cost of 0.31c. per kilowatt-hour and the cost of any more power developed would not exceed 0.5c. per kilowatt-hour.

During a period when there was considerable copper in the ore, probably resulting in a somewhat higher melting cost than usual, 2,161 lb. of dried precipitate was melted, producing 7,596.25 oz. of bullion. In the following comparison of costs it is assumed that the precipitate is dried in the muffle furnace before treatment in either the oil furnace or the electric furnace.

Cost of Melting Precipitate in the Tilting Oil Furnace

		Per Pound of Precipitate
691 lb. borax at 15c.....	\$103.65	\$0.048
178 lb. soda at 11c.....	19.58	0.009
6 crucibles at \$11.50.....	69.00	0.032
318 gal. oil at 25c.....	79.50	0.037
Furnace repairs.....	8.00	0.003
Labor for melting.....	60.00	0.027
Supervision.....	25.00	0.011
Total.....	\$364.73	\$0.167

Estimated Cost of Melting Precipitate in a 50-kw. Electric Furnace

		Per Pound of Precipitate
691 lb. borax at 15c.....	\$103.65	\$0.048
178 lb. soda at 11c.....	19.58	0.009
30 lb. carbon electrodes at 10c.....	3.00	0.001
1,515 kw.-hr. at 5c.....	7.58	0.003
Furnace repairs.....	10.00	0.004
Labor.....	18.00	0.008
Supervision.....	25.00	0.011
Total.....	\$186.81	\$0.084

In making this estimate the electrode consumption is considered as 20 lb. of carbon per ton of charge, and the power consumption is taken as 1,000 kw.-hr. per ton of charge. The lower labor cost of the electric furnace is simply due to its capacity, and depends upon the power load on the furnace. An electric furnace of smaller size might melt at the

same rate as the oil furnace when the labor cost would equal that of the oil furnace, but there would still be a considerable saving in other items.

These figures show a saving of 8.3c. per pound of precipitate melted in favor of the electric furnace, or a saving of 50 per cent. The main saving is in the source of heat and the cost of crucibles as compared with electrodes. Electric heating costs 0.3c. per pound of precipitate and oil heating costs 3.7c. per pound. Electrodes are 0.1c. per pound of precipitate and crucibles 3.2c. per pound.

For small mills where direct melting is the most feasible process, it appears that the electric furnace will melt more cheaply than the oil crucible furnace, at least where the cost of electric power is not exorbitant and fuel oil is expensive. With oil at 5c. per gallon and power at 3c. per kilowatt-hour, the melting cost in the oil-fired furnace is 13.7c. per pound and in the electric furnace 10c. per pound showing that there is application for the electric furnace in regions not isolated from the source of oil.

It should be possible to keep the losses in the electric furnace as low as in a combustion furnace. The slag loss would be as low, and experience has shown that with proper manipulation it is possible to operate an electric furnace so as to keep down volatilization to a low figure. There would be no gold loss, and silver loss should not be higher than in the combustion furnace.

R. S. WILE, Pittsburgh, Pa. (communication to the Secretary*).—H. R. Conklin of the Lluvia de Oro Mining Co., Lluvia de Oro, Chihuahua, Mex., was the first man to attempt the electric smelting of gold zinc-dust precipitates, after investigating two furnaces I had in operation at Connellsville in the fall of 1910. He published an article in the *Engineering and Mining Journal*, 1912, covering the operation of this furnace. The furnace that Mr. Lass describes is the duplicate of this one, showing no points of originality.

* Received Sept. 3, 1915.

Some Problems in Copper Leaching

Discussion of the paper of L. D. RICKETTS, presented at the San Francisco meeting, September, 1915, and printed in *Bulletin* No. 100, April, 1915, pp. 711 to 737.

FREDERICK LAIST, Anaconda, Mont.—I am going to tell you of some of the results we have been getting in Anaconda during the past two or three months in a large plant which was finished in May. We commenced work in Anaconda a year and a half ago on the construction of a leaching plant which was to have a capacity of 2,000 tons per day. The plant was to treat tailings from the concentrator, containing about 12 lb. of copper to the ton. During the past year we changed our plans and decided to use the flotation process on the tailings from the concentrator, which contained no oxidized copper, and apply the leaching process only to tailings which had accumulated in a large dump below the works. This dump contains about 15,000,000 tons of tailings, and since some of the copper in it is oxidized, we felt that the application of the flotation process would result in the loss of too much of the oxidized copper, particularly as time went on. In starting up we had a number of difficulties to overcome, as is usually the case. They were not of a serious character. The results obtained, so far, have been decidedly encouraging.

During July the plant treated a total of 50,000 tons of tailings, or an average of 1,610 tons per day. The extraction was about 83 per cent., which, on material of the grade treated, meant a tailing assaying under 0.1 per cent. copper. The acid consumption was about 65 lb. per ton, and the fuel percentage in roasting was about $4\frac{1}{4}$. We expect to cut the fuel percentage in the next month or two to about 3 per cent., the high percentage during July being due to starting up and shutting down furnaces.

We find that iron builds up in the solution to a marked degree. This is not a serious matter with us, because we have cheap sulphuric acid and can reject solution whenever it gets too foul.

The extraction of silver has been remarkably good in the large plant. It has averaged better than 75 per cent. It seems that so long as we maintain salt in the solution to a percentage of 10 per cent., the extraction is good. We allowed the salt to fall off in order to find how low a percentage we could use. At about 7 per cent. the silver extraction fell to 65 per cent. This was to be expected, because silver chloride is soluble in strong brine, but not in weak brine.

T. T. READ, Engels, Cal.—Concerning the use of SO_2 as a depolarizer, it has occurred to me, though I have not been able to try it as yet, that a hollow porous carbon anode could be used, and the SO_2 could be made to pass outward through its pores, meeting the liberated oxygen on its

outside and concentrating the action at the point where it is needed, instead of attempting to disseminate the SO_2 throughout the electrolytic bath, with the difficulties that produces. By using a sufficient pressure to force the SO_2 through the pores the difference in hydrostatic pressure between the top and bottom of the bath should not exhibit itself to a noticeable degree. If waste gases were used for this purpose they would be forced through in sufficient volume to make a strong upward current along the face of the anode, thus inducing a downward one along the cathode, so furnishing the required agitation of the electrolyte. As I say, I have not yet tried this, because I have not been able to procure any hollow porous carbon that seemed well adapted to what is required.

E. H. HAMILTON, West Norfolk, Va.—We have extracted and recovered over 80 per cent. of the copper from chemical works cinder without adding any new acid.

The acid builds up in the solution and can be kept normal by adding cinder high in zinc.

The residue is sintered and used as an iron ore.

Roasting and Leaching Concentrator Slimes Tailings

Discussion of the paper of LAWRENCE ADDICKS, presented at the San Francisco meeting, September, 1915, and printed in *Bulletin* No. 104, August, 1915, pp. 1471 to 1484.

L. D. RICKETTS, New York, N. Y.—In the Southwest we have stored up millions of tons of tailings in which the copper exists partly as sulphide and partly in an oxidized condition. Mr. Addicks' first paper suggests a likely process for the treatment of such material. His second paper¹ deals with the recovery of copper from clear solutions of notable strength, and such solutions may come from the sulphating roast or direct from the leaching of completely oxidized copper ores of which vast deposits are known to exist.

What strikes me particularly about the latter paper is the keen perception shown and the clear manner in which the writer has attacked the fundamental principles of the process. He calls your attention to the fact that ferrous sulphate is necessary in the electrolyte and is an ideal depolarizer; that sulphurous acid must be used to reduce the ferric sulphate produced and thus prevent re-solution of copper at the cathode; that mechanical agitation in the cell, in order to bring the ferrous sulphate in contact with the anode and the copper in contact with the cathode, is essential to efficient work; that lead anodes are

¹The Electrolysis of Copper Sulphate Liquors, Using Carbon Anodes, published by the American Electrochemical Society.

undesirable because they do not permit complete depolarization; and that carbon anodes are ideal for the purposes. When the ferric salts were kept low by the use of sulphurous acid there was no trouble during the life of his experiments from softening and deterioration.

E. H. HAMILTON, West Norfolk, Va.—Does sulphuric acid accumulate?

LAWRENCE ADDICKS, Douglas, Ariz.—That is a relative question. We had so much alumina, and the rate of neutralization was so high, it was more a question of getting enough acid than too much. You have raised a large question. In the process we were attempting we are bound to get whatever excess acid equivalent there is, in liquors contaminated with other things where it is of no use. If we have any acid left over, there is no market for it, consequently no revenue to be derived from it, which is the whole question.

FREDERICK LAIST, Anaconda, Mont.—Mr. Addicks spoke of the difficulty of getting a strong solution of SO_2 from a weak gas, such as would be encountered in practice. I believe Doctor Ricketts mentioned that point in connection with the paper he presented on Some Problems in Copper Leaching. I think it might be of interest to give a brief outline of some of the experiments we made in Anaconda in the course of the past year and a half, on the precipitation of copper from cupric chloride solutions, by means of SO_2 . While the plan decided on for our tailings treatment uses scrap iron as a precipitant, we wished to make sure that really would be the best method. Among other methods investigated was the old Hunt and Douglas method, which consists of adding to the copper sulphate solution sufficient common salt to chloridize the copper. The copper is then precipitated by SO_2 gas in the form of cuprous chloride, which is a white insoluble precipitate containing about 65 per cent. copper.

We carried on the process in two stages. First, we dissolved sufficient SO_2 gas in the solution. Second, we heated the resulting liquor to 90°C . which caused reduction of cupric to cuprous chloride. The latter separated out in cooling. Our solution contained about 1.5 per cent. copper and required about 0.7 per cent. SO_2 for precipitation. We found it impossible to dissolve over 0.2 per cent. at atmospheric pressure when a 10 per cent. gas was used. After we started using a closed tank for the work and forced the gas in under a pressure of about 15 lb. per square inch we obtained the desired saturation without trouble. There was no trouble about getting a 10 per cent. gas from the furnace although it was necessary to take the gas from the second floor in order to avoid admixture of air entering through the feed openings. With suitable apparatus 80 per cent. of the SO_2 , or better, could be absorbed.

The point to which I wish especially to call attention is the greatly increased strength of solution it is possible to get from a weak gas by the use of moderate pressures.

LAWRENCE ADDICKS.—I have never had any difficulty with the absorption towers, nor did we have any difficulty in getting as much gas absorbed as we did in the laboratory. We put in a stand pipe, 40 or 50 ft. high, and let the liquor run through that, obtaining absolute absorption of SO_2 . While there was no increase in efficiency, we obtained what we expected from the increased pressure.

CHARLES BUTTERS, Oakland, Cal.—Do carbon anodes soften when used as depolarizing anodes in a cyanide solution?

LAWRENCE ADDICKS.—With chloride electrolysis there has never been any difficulty. It has no oxidizing effect on the carbon. In the operation you mention, the efficiency is low since a great deal of oxygen is liberated, and carbon would certainly be attacked. You must take care of the product of depolarization.

L. D. RICKETTS.—In the Southwest we have porphyries that have been subjected to intense chemical action and contain a great deal of clay. In the old days, when we used to crush to $1\frac{1}{2}$ mm., fully 50 per cent. of the product would pass a 200-mesh screen. I think such ores would give a great deal of difficulty in leaching, even after a sulphating roast, if an attempt were made to gain a strong solution of sulphate of copper, but possibly we might make the process successful if we used a large amount of solution and produced a solution weak in copper from which the copper would be recovered with scrap iron and metallized calcine. Though such a method of recovery of copper from solution is comparatively expensive, the metal would be obtained very profitably because the material to be treated is tailings that are already mined and that are comparatively rich.

Referring to Mr. Addicks' second paper, and especially referring to the use of sulphurous acid to reduce the ferric iron produced by depolarization, at plants located at points like Douglas, where immense amounts of sulphur are going to waste as sulphurous acid gas, the question of efficiency does not enter into the problem. In certain districts in the Southwest, especially at Ajo and Globe, Ariz., we would have to consider the importation of pyrite or sulphur for the manufacture of the sulphurous acid gas. In such cases the question of complete absorption becomes very important, yet complete absorption of the gas as produced by burning sulphur or pyrite presents many difficulties. In our experiments we did not get very good absorption in a coke tower. The solution possibly followed channels. At all events we found if we could spread out the liquid in very thin sheets over large surfaces we obtained much greater

absorption. We also found that a neutral solution absorbs more gas than an acid solution. Where, however, complete absorption is important, it occurs to me that the pumping cost will be considerable and the cost of the construction of the necessary absorption towers will be very great.

Mr. Addicks' paper calls attention to the fact that if solutions containing ferric salt and sulphurous acid are heated to a point about 130°F., the reduction is almost instantaneous. I think the question of obtaining a pure gas is important. If by some cheap method 100 per cent. sulphurous acid gas could be obtained, the necessary amount could be pumped under a low pressure into the warm solutions and the reduction might be instantaneous, and the cost of both operation and construction might be very considerably lessened.

T. T. READ, Engels, Cal. (communication to the Secretary*).—Mr. Addicks' paper (Electrolysis of Copper Solutions with Carbon Anodes) is extremely suggestive and helpful, and I find myself in special accord with his method of attack as I have always believed that copper hydrometallurgy would eventually develop along the lines of cyanide practice, namely, fine grinding and the treatment of such material by the methods it requires. This seems even more true now that flotation is becoming general, for fine grinding is a prerequisite to that, and it seems quite likely that practice of the future will develop along the lines of a combined flotation-hydrometallurgical process. It is fairly easy to float sulphides; it seems foolish, therefore, to roast them at considerable cost, preparatory to an acid treatment that is no more satisfactory than flotation. Oxidized minerals yield easily to acid treatment; it seems at least doubtful whether they can ever be converted into sulphides and floated, thus putting them into concentrate that requires a subsequent treatment, and still show a profit in the operation. Of course there is the advantage of such a scheme that you are getting the desired mineral in a concentrated form, whereas hydrometallurgy is fundamentally unsound in that you take ore that contains only a few pounds of copper per ton and dissolve that copper out into three or more tons of solution, thus diluting it to one-third of its original dissemination. The first step in hydrometallurgy is not, therefore, concentration, but dilution, and is fundamentally bad. On the other hand, the problems of hydrometallurgy are largely mechanical and the advantages of handling the ore in the form of pulp are so great that it seems to more than counterbalance this drawback. On the whole it seems likely that the treatment of copper ore by sulphuric acid will develop along the same general lines as cyanidation, and we will do well to direct our attention where Mr. Addicks has, to the treatment of slime tailing.

* Received Oct. 30, 1915.

It would be well, by the way, if we had a better name for this field of work. It is now commonly called copper leaching, but that is a misnomer, for the work Mr. Addicks has described does not involve leaching at all, and even where leaching is used it is not the most important of a series of operations that are necessary. Hydrometallurgy of copper is cumbersome and too general in its scope to define exactly the treatment of copper ore with sulphuric acid and the subsequent recovery of the copper from sulphate solution. It would be well if some one introduced a new word of precise meaning, just as cyanidation is perfectly definite. Possibly vitriolization would do, or perhaps some one will suggest a better word.

The Occurrence of Covellite at Butte, Mont.

Discussion of the paper of A. PERRY THOMPSON, presented at the San Francisco meeting, September, 1915, and printed in *Bulletin* No. 100, April, 1915, pp. 645 to 677.

ARTHUR S. EAKLE, Berkeley, Cal.—At the old Morning Star mine in Alpine County, where large masses of enargite were found some years ago, I looked particularly for evidences of replacement of pyrite by the enargite. While I have not made a particular study of any of the specimens, I judge that the pyrite and enargite were precipitated together. They were both primarily in those rocks. Of course, by using Mr. Graton's method of determination we might possibly find a situation somewhat different, but I should say that the enargite is not replacing the pyrite, but was simply precipitated at the same time. The bodies occurred there within 50 ft. of the surface, so it is probably not a case of secondary enrichment.

H. W. TURNER, San Francisco, Cal.—In the case where secondary chalcocite shows a crystalline structure and cleavage lines, what is the evidence of the secondary nature of it?

C. F. TOLMAN, Stanford University, Cal.—You can find practically all the evidences we have discovered in the microscopic study of ores. You can find all stages in the process of replacement of earlier minerals, especially bornite, by chalcocite, from a stage, in which the process has but just started with only occasional fragments of chalcocite set in bornite, to a stage where remnants of bornite are set in chalcocite. The replacement may take place along the cleavage lines or along the crystallographic directions of the earlier mineral, or it may take place irregularly. In one type of replacement you may find in one spot a few cross-cutting veins, and in another spot the latter may be so enlarged that there remain only a few individual specks of the replaced mineral.

A. C. LAWSON, Berkeley, Cal.—I would like to ask whether this structure is really a pegmatite structure.

C. F. TOLMAN, Stanford University, Cal.—It is the so-called eutectic structure of Laney and others.

A. F. ROGERS, Stanford University, Cal.—In some cases the so-called graphic intergrowth of chalcocite and bornite is probably due to inherited structure from an earlier graphic intergrowth of bornite with other minerals. In other cases it seems to be due to a partial breakdown of bornite into the chalcocite. I have not found any case where bornite and chalcocite were formed simultaneously. I may be mistaken but I do not believe that they were formed simultaneously from ascending solutions. The so-called intergrowth is a peculiar breakdown but it is rather difficult to know just what it means.

L. C. GRATON, Cambridge, Mass.—There was, as Professor Tolman has said (in his summary), an epoch-making paper on the application of the reflecting microscope to the study of sulphide ores, but that was the paper by Campbell. Similar methods had been employed many years earlier, but Campbell was the first to disclose the general applicability and importance of the idea and most of the progress that has been made in this direction is traceable directly or indirectly to the impetus he gave to this line of microscopic research. Murdoch and I were simply among the early followers of Campbell's methods, and if our paper contained any contribution it lay chiefly in our attempt at systematic attack on a single part of the entire field this new method opened up.

Apparently Mr. Thompson, and perhaps some others, have fallen into an error for which Murdoch and I may have been responsible, viz., the conclusion that a systematic pattern developed in chalcocite by etching with acid (cleavage structure, if you will), is a certain indication that the chalcocite was formed from ascending solutions, i.e., is "primary" in the generally accepted sense. Shortly after the appearance of our paper, Professor Tolman pointed out the danger in applying this conclusion too sweepingly, as we had likewise cautioned, but Mr. Thompson seems to have overlooked these warnings. There can be no doubt that under certain conditions a systematic structure revealed by etching is present in chalcocite that has been formed by descending solutions in the orthodox, secondary-enrichment fashion.

The question of the "graphic intergrowths" affords another topic for discussion. Laney was the first to note in the sulphides that peculiar pattern with which we are familiar in the pegmatites, and he concluded that it meant, in the one case, as had commonly been regarded in the other, simultaneous crystallization of the components. At the time our paper was published Murdoch and I had found no reason to dispute this

interpretation, and I am by no means ready as yet to accept the extreme belief expressed by Professor Rogers that this structure never represents simultaneous intergrowth, though I am now satisfied that a very similar pattern may in certain cases result from partial alteration of one mineral to another.

Criticism of the terms "primary" and "secondary," particularly the latter, on the ground that they have more than one meaning, is to some extent justified. As Professor Tolman has indicated, the terms "hypogene" and "supergene," proposed by Ransome, are valuable because of their precision as to direction from which the originating causes were derived. But the term "secondary enrichment" has been so extensively used and has come to mean so definite a thing, and to be understood so unusually well by all classes of people engaged in or connected with mining of the metals, that it would seem to be as futile as it would be unwise and confusing to endeavor to abandon it and substitute some other expression for it.

Equally to be deplored and avoided, it seems to me, is the application of the term "secondary enrichment" to results and processes that have little or nothing in common with those alterations by descending, meteoric waters that produce an increase in metal content of part of a deposit through the impoverishment of a nearby part. Coupling "upward" or "ascending" with "secondary enrichment" is in my opinion a mistake.

A. C. LAWSON.—One matter that appealed to me in connection with this discussion is the desirability of not disturbing the nomenclature after it has once been established in the mind of the layman or practical miner. If we change our nomenclature we will disturb people and confuse them greatly, and in fact we will undermine the confidence they have in our work. Whatever proposals are made for the change of the use of words such as "secondary" and "primary," we should always fully comprehend the meaning attached to those words by the miners who have come to use them. Those words to them have a certain definite meaning. I agree with Mr. Graton in the hope that the term "secondary" be retained, but that we should all have a definite understanding of the meaning of secondary enrichment. If the idea has prevailed that primary ores were all formed at one time I am very glad the trend of the discussion shows that that idea has been broken up.

J. C. RAY, Palo Alto, Cal.—I think I am probably more interested in Mr. Thompson's paper than any one else present, as I have written a paper on the same subject which appeared last year in *Economic Geology*. It naturally gives me no little satisfaction to find that Mr. Thompson departs rather cautiously from Sales' theory, and to note his practical verification of the principal points which I have already brought out;

i.e., that the Butte deposits were formed as the result of several distinct periods of mineralization whereby each set of sulphides belonging to one period was replaced by a later set of sulphides belonging to another period, and not, as thought by Sales, by a practically continuous circulation of a gradually changing solution which resulted in the simultaneous deposition of several sulphide ore minerals. I make this statement reservedly as I know that Sales wrote his paper before any detailed study of the Butte ores had been made with the use of the reflecting microscope, and presume that he made statements then which he is willing to modify or change now.

The soundness of Mr. Thompson's criteria for distinguishing primary from secondary chalcocite by the etch structure has already been questioned. I also do not agree with Mr. Thompson in this point for I have seen much undoubted secondary chalcocite which exhibits the same regularity of etch structure which Mr. Thompson ascribes to primary chalcocite.

He also believes the transitional zones to be indicative of secondary or downward solutions. My own observations would lead me to place equal weight with this as evidence of minor side reactions which frequently accompany the replacement by ascending solutions. While undoubtedly taking place at times from the action of descending solutions I can not agree that it is a safe criterion for downward enrichment.

Mr. Thompson frequently refers to the replacement of quartz, pyrite and quartz-monzonite by primary chalcocite. I can not agree with this. I believe that any primary (deposited by ascending solutions) chalcocite is a replacement of earlier sulphides, enargite or sphalerite, usually through the intermediate stage of bornite. I believe that the replacement of pyrite by primary chalcocite is of little importance. The author mentions chalcocite occurring in veinlets up to 2 ft. in width which are direct replacements of quartz, pyrite, or monzonite. I have studied rich chalcocite ore from some of these little veins and find the chalcocite undoubtedly replaces earlier sulphides. One such, the south branch of the Minnie Healy, on the 1,600-ft. level of the Tramway mine, exhibits ore which is apparently pure chalcocite replacing pyrite and the monzonite wall rocks. Microscopic study shows this ore to be thickly dotted with residual fragments of enargite and bornite with occasional remnants of sphalerite. Some of this chalcocite is completely meshed with partially replaced enargite grains. At no place in Butte have I been able to find primary chalcocite which I could conscientiously ascribe to anything but a replacement of enargite, sphalerite, or covellite. The evidence is not always clear in one specimen, but by studying many specimens the solution of the replacement is sure to be found somewhere.

L. C. GRATON, Cambridge, Mass.—Do you exclude the bornite?

J. C. RAY.—By no means. In the area which is under discussion in Mr. Thompson's paper, chalcocite replaces enargite directly, but I am inclined to think that the greater part of it has gone through the bornite stage first.

H. W. TURNER.—I quite agree with what Professor Lawson has said in regard to retaining the terms "primary" and "secondary" in the original sense used by S. F. Emmons for the use of the mining fraternity. There is also a financial point in this. Investors in sulphide mines the world over have come to know about primary and secondary ores, used in this simple sense; and they are interested because if a sulphide deposit has been leached above, it is nearly certain that some rich ore will be found below, unless the primary sulphides were merely barren pyrite. Even the intelligent miner now understands the significance of the two terms.

The distinction between the various zones in copper mines, in which the original minerals were chiefly pyrite and chalcopyrite, is very plain; but it is true that in complicated deposits, like those of Butte, the average miner, or even mining engineer, would not be able to draw altogether correct conclusions as to the limit of the zone of secondary enrichment and the primary zone below without the help of such investigations as those at Butte, by Emmons, Weed, Sales, Graton, Tolman, Rogers, Ray, Thompson, Simpson and others. There was "upward sulphide enrichment" in the primary zone; and the original solutions may have reacted upon each other resulting in the replacement of one sulphide by another; yet all these reactions belong to the primary period, long antedating the later leaching and secondary sulphide enrichment period.

The scientific description and discussion of the reactions and replacements that took place during the primary period requires a special set of terms with definite meanings, but it is not necessary that the layman should be required to know them. This will be all the more true if investigators always add a chapter to their papers, in which their general conclusions are stated in plain language, and in which, in reference to ore deposits, the terms "primary" and "secondary" are used in their original sense. Moreover, the writer of a special paper owes it to the public, as well as to himself, to add such a special chapter, for, being more familiar than anyone else with the subject in hand, he can best do this.

Of paramount importance to the mining industry are the broad facts, in most sulphide deposits, of (1) a leached zone, (2) a zone of secondary enrichment, (3) a zone of primary ore, and the terms "primary" and "secondary" should be retained in this sense and so used not only in popular papers, but in scientific discussions in order to avoid confusion.

A. F. ROGERS.—We are accustomed to the use of the terms "primary"

and "secondary" in a geological sense, but if the matter were looked into carefully, the word "secondary" would probably be found to have priority in a mineralogical sense. At any rate, the terms primary and secondary as used at present are ambiguous. Some of us use them in one sense and some in another. The difficulty is that sometimes in the same paper the author will use the terms in two different senses, so it seems to me that in a paper it is absolutely necessary to define the use of these terms. For my part, I should like to avoid the use of these terms altogether, but that is a difficult thing to do. We could use such terms as "the first generation" and "the second generation." I favor the use of the terms "hypogene" and "supergene," as indicating the source of the solution.

The terms "primary" and "secondary" may then be dispensed with and in their place the words "hypogene" and "supergene" used. The point is that these terms indicate the source of the solutions, and that is the principal argument for their use.

C. F. TOLMAN.—There is a difficulty in regard to the present use of the terms "primary" and "secondary," which apparently has not been fully appreciated by the preceding speakers, but which is felt by those of us who are engaged in the microscopic study of ores. This difficulty is due to the fact that a genetic-geological meaning has been grafted on to the purely descriptive terms for those minerals formed at the expense of earlier sulphides. These terms must be divorced from any theory as to the origin of the ores or their structures; e.g., I want to describe a certain ore, which is, say, a mass of bornite shot through with chalcocite stringers. I know that the bornite is being broken up and replaced by chalcocite, but I do not know anything as to where the solutions causing this change come from; whether they come up or down or sidewise or any other direction.

Doctor Rogers suggests that we say chalcocite of the first and second generation. The difficulty here is that the first generation of some one mineral, say chalcocite, may be secondary in and derived from some earlier mineral, say bornite, and chalcocite of a second generation may be deposited in veinlets and therefore be primary in the mineralogical sense. In either case the chalcocite may be of the fourth, fifth or n th generation, if we broaden the term and consider each step or stage in the formation of minerals of the ore as a generation. Any of these different generations of minerals and stages of mineralization might be produced by ascending or descending solutions. In the microscopic study of an ore, we must have a set of names for those minerals that are formed at the expense, and by the destruction of (partial or complete) an earlier mineral or minerals. For such minerals geologists have used the term "secondary," long before the process of downward enrichment was dis-

covered or the phrase "secondary sulphide enrichment" was invented. In the early usage, secondary carried no implication as to whether the solutions causing the alteration were hot or cold, ascending or descending. For example, lepidolite or muscovite develops along tin veins, as a secondary mineral. The solutions causing this alteration are probably in gaseous form and generally ascending. Two common secondary products in the alteration of feldspar are sericite and kaolin. The former is probably generally produced by ascending solutions and the latter by descending solutions. Mr. Turner's suggestion, therefore, that we retain the terms "primary" and "secondary" in the *original* sense used by S. F. Emmons is somewhat confusing.

The speakers seem to be generally of the opinion that priority should not rule in this case; that secondary should carry with it the concept of oxidation producing a leached zone, of downward moving solutions, of wealth at or near the water level, of poverty with depth; to be, in short, a name for stock jobbers and engineers to conjure with, to entice the fearful or to frighten the sanguine. Since language is a living thing, ruled by force and necessity and not by logic and priority, these terms may finally acquire the burden of geological theories. If so, we must have a set of names for the microscopist. We must understand each other. Loose language produces loose thinking. Think of the harm to which the present confusion in nomenclature has given rise. Professor Krusch has said that he has never seen primary chalcocite. (He uses the term in its prior meaning.) His critics cited the "primary chalcocite" in Butte. This turns out to be secondary as Professor Krusch used the term, and only by a process of geological theorizing can it be called primary.

At the meeting of the Institute in February, 1914, there was a discussion¹ of the question "To What Extent is Chalcocite a Primary, and to What Extent a Secondary Mineral in Ore Deposits?" In this discussion J. D. Irving appears to use secondary in the geological sense. F. L. Ransome avoids the use of the terms and his remarks are not ambiguous. Waldemar Lindgren uses the term in the mineralogical sense speaking of primary chalcocite crystals, although taking part in a discussion in which most of the disputants are using the geological or a mixed meaning. L. C. Graton frankly assumes that if a mineral is secondary in a mineralogical sense, it therefore is formed by the processes of downward enrichment. It was this assumption that so greatly diminished the value of his epoch-making and otherwise valuable paper on The Sulphide Ores of Copper.²

It seems necessary, somehow, someway, to adopt a nomenclature accurately defined and understood by all. If, however, the struggle

¹Trans., xlviii, p. 194 (1914).

²Trans., xlv, p. 26 (1913).

is to continue and a peace conference is not as yet in order, let each man state clearly the way in which he uses the terms "primary" and "secondary."

A. PERRY THOMPSON (communication to Secretary*).—Among the most interesting details of the study of opaque minerals in the sulphide ores are the criteria for distinguishing the primary from the secondary minerals and the relation of bornite to chalcocite. In the discussion, the statement was made that a regular cleavage or pattern developed in chalcocite by etching with acid is not an unquestionable indication that the chalcocite was formed by ascending solutions. In this I fully agree, as perfect cleavage structure developed by etching chalcocite deposited by descending waters has been observed in many Butte ores since the writing of my paper in March, 1914. In April, 1915, in a communication to the Editor, I attempted to bring the manuscript up to date and corrected the former conclusion that the regular cleavage lines developed by etching are a reliable criterion for the determination of primary chalcocite. Unfortunately, this correction arrived too late to be made in the paper.

The relation of bornite to chalcocite is difficult to understand at all times, particularly by means of the microscope alone. Most commonly the relation may be described, as Professor Rogers has said, as a peculiar breakdown of the bornite in the chalcocite. Tolman and Clark, combining an intimate knowledge of the ores through microscopic study with chemical investigation, have lately published an instructive paper on the formation of the sulphide ores, particularly those related closely to chalcocite. More recently in the *American Journal of Science*, June, 1914, E. H. Kraus published the results of chemical analyses of a number of minerals grading in chemical composition, in orderly fashion, from bornite to chalcocite. Many of the minerals described by Doctor Kraus are probably the opaque minerals observed under the microscope, having slightly different shades, but closely resembling bornite in all characteristics, that have puzzled the microscopist and have been referred to as products of the general breakdown of bornite.

* Received Oct. 18, 1915.

The Geology of the Iron Ore Deposits In and Near Daiquiri, Cuba

Discussion of the paper of JAMES F. KEMP, presented at the San Francisco meeting, September, 1915, and printed in *Bulletin* No. 105, September, 1915, pp. 1801 to 1836.

WALLACE E. PRATT, Manila, P. I.—I am glad that Professor Kemp has referred to the Philippine ores in his paper, inasmuch as I have had the privilege of examining, in a preliminary way, the ores there. I must say, from the résumé given us, that the Daiquiri ores seem to be similar in origin and character to the magnetite-hematite ores in Bulacan Province, Philippines.

Professor Kemp has also referred to C. M. Weld's description of the magnetite-hematite ores at Hongkong and to the similarity of these ores to the ores at Daiquiri and to those in Bulacan. There is a minor feature of apparent difference in origin in the Hongkong and Bulacan ores. Mr. Weld believes the iron ores at Hongkong to have resulted from the intrusion of the Hongkong granite, a distinctive rock of wide distribution in the Orient, into sedimentaries which are probably of much greater age than those which have been replaced in part by the ores in Bulacan. Associated with some of the ores in Bulacan and elsewhere in the Philippines there is a granite which may be the equivalent of the Hongkong granite; but the Bulacan ores are not related in origin to the intrusion of this granite, which is clearly older than the ores, but to the later intrusion of dikes and small bodies of basic rocks into the granite and into associated tuffs, breccias, and sedimentaries, including limestone. I gather that the intrusives to which the mineralization is related at Daiquiri are also small in volume and later than larger bodies of associated igneous rocks, from which it would appear that the Bulacan ores are even more closely related to the Daiquiri type than are the Hongkong ores.

The deposits in Bulacan are inaccessible and are not large, the largest one containing about 1,000,000 tons, according to an estimate based solely on its outcrop dimensions. Ores of similar character, however, occur in Camarines Province in apparently larger quantity and in a situation on the coast adjacent to deep water.

The Copper Deposits of San Cristobal, Santo Domingo

Discussion of the paper of THOMAS F. DONNELLY, presented at the San Francisco meeting, September, 1915, and printed in *Bulletin* No. 104, August, 1915, pp. 1759 to 1768.

F. LYNWOOD GARRISON, Philadelphia, Pa. (communication to the Secretary*).—The paper is of value in giving the public some first-hand information regarding a copper property that from various causes has attracted considerable attention, especially in Philadelphia, and which has also been the source of several fantastic flotations that had little to recommend them.

Since 1906, no work has been done on this property under my direction, with the exception of a topographic survey made last winter by Mr. Donnelly, which was intended as the first step in a sound and logical system of development. From 1906 to 1915, desultory attempts were made to develop these orebodies, but as the work was not in competent hands much money was wasted and even more disappeared in ways regarding which it is perhaps not best to inquire. The result is, this interesting property has now practically no more ore developed upon it than in 1906, despite the fact that a new 100-ton mill is being erected, although, as I have intimated, it is today not much more than a prospect.

Turning now to the scientific and more agreeable side of the subject, it is interesting to note that these Santo Domingo copper ores are invariably found in volcanic tuffs and that, in a geologic sense, they are all very old, for there are no evidences of recent volcanic activity in the greater Antilles, of which this island is the second largest. The existence of copper in tuffs in different parts of the world is a notable fact. Often, if not usually, it is present in mere traces, but not infrequently we find it in sufficient amount to be of economic importance. Thus, for example, chalcopyrite and bornite occur in the Tertiary tuffs of Japan, and a number of producing copper mines of that country are either actually in, or closely associated with, rocks of this character. Whether the copper minerals are original primary ores in the tuffs, or were derived from accompanying eruptive dikes, appears not yet to have been determined in any instance. In most of the Japanese cases of which I can find any record the tuffs appear to be associated with shales and limestones. These Japanese copper ores also carry some gold, especially in the upper parts of the veins; in this they resemble the Santo Domingo deposits which not infrequently carry as much as \$6 or \$7 per ton of gold on the outcrop. As a rule the content of gold in both the Japanese and

* Received June 29, 1915.

Santo Domingo deposits decreases in depth very rapidly, and a few feet down the average tenor appears to be about \$1.50 per ton. The explanation is simple; the copper minerals being more soluble they are leached out on or near the surface leaving the gold behind. It is not, however, so easy to account for the fact that visible gold is often present in the outcrop oxidized ores and never perceptible in the sulphides. The only explanation that offers itself for this peculiarity is that the gold may in some manner be dissolved and then reprecipitated by the ferrous sulphate certain to be present in waters flowing through mineralized and jungle-covered rocks.

The Braden Copper mines in Chile are reported to be in the perimeter of a volcanic vent filled with tuffs. The copper sulphides and metallic copper occur in seams and veinlets in diorite and brecciated contact material.¹

It is a notable fact that of late years volcanic tuffs are becoming more and more to be recognized as the habitat of metallic minerals and ores. Rocks of this character are not always readily recognized as such, being often denominated andesite, olivine-basalt, diabase, or the general term basic eruptives. In view of the fact that a real tuff reaches the surface of the earth as a volcanic ejectment in the form of dust (sometimes termed ash or mud) to be consolidated afterward by surface phenomena, whereas effusive rocks such as andesite, diabase, diorite, etc., appear as magmas or lavas that have flowed out from the interior of the earth, this confusion of terms is unfortunate and bespeaks a carelessness of observation and diagnosis. It is evident on the whole that copper minerals have an affinity for tuffs which must be recognized, and it may not be going too far to assume that such volcanic dust is the original source of the copper we find in the fissures of the rocks which have resulted from the consolidation of these ejectments.

¹ *Engineering and Mining Journal*, vol. lxxxiv, p. 1060 (Dec. 7, 1907.)

The Formation and Distribution of Bog and Iron-Ore Deposits

Discussion of the paper of C. L. DAKE, presented at the San Francisco meeting, September, 1915, and printed in *Bulletin* No. 103, July, 1915, pp. 1429 to 1436.

A. C. LAWSON, Berkeley, Cal.—It appears that our bog-iron ores are due in a large measure to the activity of certain iron bacteria, and a paper dealing with this subject that makes no reference to that particular agency strikes me as rather curious.

C. L. DAKE, Rolla, Mo.—Mr. Lawson's criticism that this paper makes no reference to the activity of iron bacteria is scarcely true. On p. 1430 I stated: "Certain bacteria precipitate iron by building the hydrated oxide into their body cells," and again, "There are many other means by which, under special conditions, iron may be precipitated in nature. Oxidation, evaporation of CO_2 , and iron bacteria have been held to be the most important." Thus the activity of bacteria receives as extended mention as the action of any other agency in precipitation of iron. The paper makes no pretense of covering exhaustively the origin of bog ores, but, as its original title suggested, refers to Some Principles Controlling the Formation and Distribution of Bog Iron Ores. Mr. Lawson seems to have entirely missed the main thesis of the paper, which attempts to establish some relation between glaciation and abundance of bog ores now forming.

Unfortunately, I have no immediate access to the literature of the subject, but my best recollection is that the data regarding iron bacteria throw no light whatever on the cause of bog ores being so much more abundant within the limits of the glaciated regions, and being reported much less frequently from the tropics.

This lack of data is suggested in the following (p. 1434): "It has already been shown that bacteria aid in the precipitation of iron. Whether tropical conditions increase or decrease the activity of such bacteria is unknown."

Since I had no data showing that iron bacteria were in any way responsible for the peculiar distribution of bog ores, further reference to them in the paper cited was deemed to be extraneous to the subject.

G. H. Cox, Rolla, Mo. (communication to the Secretary*).—What appeals to me as being the most interesting and important part of the paper is the conclusion that glaciation has in general been the chief factor in the formation of bog-ore deposits. So far as I am aware, this, if true, constitutes an addition to our knowledge of the subject.

* Received Oct. 9, 1915.

The paper with its references will be a valuable starting point for one interested in the subject, and it is to be hoped that it will increase the interest in meteoric-iron deposits so that satisfactory explanations for the solution and transportation of such quantities of the apparently almost insoluble iron oxides and the precipitation of this material as compounds varying in composition from amorphous iron hydroxide to crystalline blue hematite will be forthcoming.

Geology of the Burro Mountains Copper District, New Mexico

Discussion of the paper of R. E. SOMERS, presented at the San Francisco meeting, September, 1915, and printed in *Bulletin* No. 101, May, 1915, pp. 957 to 996.

A. C. LAWSON, Berkeley, Cal.—One point to which Mr. Graton (in summarizing the paper for us) called attention is that the author had not explained why some copper was fixed in the zone of oxidation. I recollect very well puzzling over that question at Ely, Nev., some years ago. Mr. Sales, in his paper on Butte,¹ suggests the explanation. It is due according to him to the paucity of pyrite in the primary ore. Where there is abundant pyrite sufficient sulphuric acid is formed by oxidation to take all the copper into solution and carry it to lower levels; but where the pyrite is scant so that insufficient sulphuric acid is formed, some of the copper is carbonated and so fixed in the gossan. The hypothesis, or some modification of it, as for example the combination of the sulphuric acid with lime, would appear to explain satisfactorily the fixation of the copper in the gossan as carbonate. I have, however, had no occasion in recent years to study the phenomenon.

L. C. GRATON, Cambridge, Mass.—As I remember, the rocks in which the chrysocolla occurs in the Burro Mountains are both silicified and kaolinized.

It seems probable that in surroundings deficient in carbon dioxide, complete oxidation of chalcocite may yield chrysocolla up to a maximum of half the copper in the chalcocite; while in rocks, like garnet contact zones and certain easily decomposable porphyries, surface decomposition appears often to yield silica in the necessary form to convert the copper of primary sulphides into chrysocolla immediately and directly upon oxidation.

¹ Reno H. Sales: Ore Deposits at Butte, Mont., *Trans.*, xlii, p. 49 (1913).

Ventilation of the Copper Queen Mines

Discussion of the paper by CHARLES A. MITKE, presented at the San Francisco meeting, September, 1915, and printed in *Bulletin* No. 105, September, 1915, pp. 1941 to 1958.

GERALD SHERMAN, Bisbee, Ariz.—Mr. Mitke justly might have made much stronger his claim for increase in efficiency of labor due to the introduction of forced ventilation. The product per man shift in stoping in the Gardner division has practically doubled. While it is true that improved methods have been responsible for a great part of this gain, and what is due to better ventilation cannot be definitely known, yet it is also true that the results could not have been obtained in that division as it was in 1912.

The paper discusses the two systems of ventilation in use, exhaust and pressure. In both the object is the same, to take foul air or gas out of the mine by the most direct course and replace it by fresh. In coal mines, the exhaust system is in almost universal use, and since the most advanced work in ventilation has been in that class of mines, it was naturally the first one to be considered. It has the advantage of leaving all adits, shafts, or other openings free for use. Doors are only needed to direct the course of the air, and not to confine it.

Its disadvantages appear in case of fire, or if the ground over the mine is much broken by subsidence. The fan must work in gas in the case of a fire, and may be damaged or destroyed either by heat or by corrosive gas. At the time of the fire in the Lowell division of the Copper Queen, a small exhaust fan was placed over a ventilating raise, but the vanes and housing were destroyed by sulphurous gases within a few weeks.

If the ground above the mined area is much broken, as is frequently the case when the ore deposit is thick vertically, a good deal of air is likely to pass through it, particularly if it is near the surface. Leakage from this source is a loss to the exhaust system by short circuiting, which is exaggerated by a contraction or block in the air course. It serves as an added outlet in the pressure system. This is strikingly shown at the United Verde, where many jets of steam may be seen coming up through the ground above the stoped country on a cold day.

If there is a fire, a cave in the air course between it and the exhaust fan would be a serious matter, while in the pressure system, the gases might be forced out through the broken ground, and the pressure could be raised if necessary to do so.

Mr. Mitke briefly mentions that repairs were made in the Lowell shaft by men using oxygen helmets. I believe that members of the Institute would be interested to hear a little more of this work.

In the Lowell fire of 1911, the caving of an air course diverted sul-

phurous gas to the shaft at the 800 level, through which it passed to the surface. As all mining was below this level, workmen entered the mine from other shafts, but for some little time cages were run through the gas, carrying waste rock, timbers and other supplies. The lag screws, pipe and cable hangers wasted rapidly by corrosion, the guides loosened and both cages finally jammed, although they were afterward pulled out. Mr. Mitke was borrowed from the Stag Canon Fuel Co. at Dawson, N. M., one of the Phelps, Dodge properties. He trained a crew of miners in the use of oxygen helmets and, by working from the cages, secured and replaced guides, etc., cut out dead power cables and generally repaired the shaft through the gas. It took three weeks of work, in two shifts, with frequent relays of men. That it was done successfully, and without accident, is due to Mr. Mitke.

JOSEPH P. HODGSON, Bisbee, Ariz.—Since artificial ventilation has been installed at the Copper Queen mine, the benefits have been so great for both the men and the property that, while other things have entered into the problem besides ventilation and changes in mining methods, we are anxious and willing to have a large share of the credit bestowed on the ventilation system. It was adopted after careful thought and much study, and we are well pleased with it. If we were to extend our ventilation system, or to put one in anywhere else, we would certainly install the system we have adopted, as it gives a large amount of air per man per minute, and we have the mine at all times under pressure so that we can control it in case of fire.

Underground Mining Systems of Ray Consolidated Copper Co.

Discussion of the paper of LESTER A. BLACKNER, presented at the San Francisco meeting, September, 1915, and printed in *Bulletin* No. 102, June, 1915, pp. 1249 to 1290.

SIDNEY J. JENNINGS, New York, N. Y.—In the historical part of the development of the caving system discussed by Mr. Blackner, I notice the absence of one man's name, Charles Henroten. Mr. Henroten used the caving system in South Africa in the Kimberly mines, and realized from his work there that ground would cave when properly prepared. I think it is due him that his name should be mentioned in connection with an exposition of this system in America.

The Advantages of High-Lime Slags in the Smelting of Lead Ores

Discussion of the paper of S. E. BRETHERTON, presented at the San Francisco meeting, September, 1915, and printed in *Bulletin* No. 104, August, 1915, pp. 1595 to 1599.

ANTON EILERS, Brooklyn, N. Y. (communication to the Secretary*).

While the advantages in the use of high-lime siliceous slags in lead smelting which Mr. Bretherton discusses in this paper are obvious under certain economical conditions, he gives me credit for the introduction of such slags when it does not belong to me. It is true I did considerable experimental work, off and on, for the purpose of finding such slags, while at the Germania in 1877 and 1878, but I did not have occasion to introduce any such slags for regular work, because I had no ores which fitted the economical conditions for using high silica and high lime, except in isolated cases for very brief periods. On the contrary, the ores I had to treat were mostly very ferruginous and very low in silica, so that, economically, I was compelled to run nearly always on slags, which allowed me to use the abundant flux I had without cost.

It was August Raht who first had occasion to establish and use for long periods a slag high in silica and lime, and low in iron. This was when he was confronted with the task of smelting the Horn-Silver ores at the smelting works at Franklin, Utah, with as little expenditure for fluxing material as possible. I was at that time in Leadville, Col., where, in the new works I had built, the Billings and Eilers smelters, I had occasion to use profitably many slags, but never anything as high in the earthy bases and in silica as Mr. Raht had to use.

On Dec. 4, 1881, Mr. Raht sent me the following two analyses of slags, taken from the regular run on different days, and the remarks accompanying:

	I Per Cent.	II Per Cent.
SiO ₂	33.9	35.0
FeO.....	26.8	25.4
CaO.....	26.1	24.9
BaO.....	Not det.	3.2
Al ₂ O ₃	10.8	9.3
	97.6	97.8
Pb.....	0.6	0.8
Ag.....	0.0	0.2
I. Produced with coke = 15.2 per cent.		
Bullion 163 oz.; lead contents of charge, 18.7 per cent.		
Blast pressure, 5½ oz. per square inch.		
II. Produced with coke = 14 per cent.		
Bullion 152 oz.; lead contents of charge, 20.1 per cent.		
Blast pressure, 6¾ oz. per square inch.		

* Received Sept. 8, 1915.

This was a remarkably clean slag and, according to Mr. Raht, kept the furnaces in excellent condition and allowed of good tonnages. I looked up the nearest chemical formula that I could derive from the above percentages and found the following:



Assuming that the slag consisted only of silica, iron protoxide, and calcium oxide, it gives the following percentages:

SiO ₂	35.28
FeO.....	31.76
CaO.....	32.94
	99.98

and assuming that 12 per cent. of iron and lime was replaced by foreign bases (in this case principally Al₂O₃ as given by Mr. Raht) the following percentages result:

SiO ₂	35.28
FeO.....	25.87
CaO.....	26.83
R.O.....	12.00
	99.98

I therefore took this slag of Mr. Raht's to be a "correct slag" with all the virtues that correct slags have.

It will be noticed that it is a singulo-silicate of iron and lime plus a bisilicate of iron.

I occasionally used this slag, later in life, with advantage in both lead and copper smelting.

Mr. Bretherton also kindly credits me with the introduction of cast-iron water jackets. Cast-iron water jackets closed on top with an outlet pipe for the hot water on the side near the top (the inlet pipe for the cold water being near the bottom of the jacket) were in use at a few smelting works when I came to Salt Lake City, but frequent burning out on account of steam spaces forming, made their use rather costly. Together Augustus Steitz, of St. Louis, and I devised a jacket with an open neck 7 in. high, of box shape, on the top of the jacket, into which the cold-water pipe reached down 10 or 12 in. and the hot-water outlet pipe was very near the top of the neck, thus preventing any steam spaces in the jacket. It is true this jacket was generally introduced where cast-iron jackets were used. Some people, however, used costly wrought-iron jackets in preference, the reason for which has never been apparent to me.

The advantages which Mr. Bretherton sets forth, derived from the use of high SiO₂ and high CaO slags in general, coincide with my own

observations. Of course, the main reason for using such slags must always be the economic side in the selection of flux, but the other incidental advantages which he mentions are also of great moment. The elimination of arsenic in certain charges can, according to my experience, be increased by cutting the fuel percentage slightly below that needed for perfect reduction, and by giving, at the same time, say, 1 per cent. more CaO than the formula requires. This, however, may increase the silver loss slightly.

Some Suggestions Regarding the Determination of the Properties of Steel

Discussion of the paper of A. N. MITINSKY, presented at the San Francisco meeting, September, 1915, and printed in *Bulletin* No. 104, August, 1915, pp. 1697 to 1705.

LAWFORD H. FRY, Burnham, Pa. (communication to the Secretary*).
—The author assumes that the proportional limit and the elastic limit are coincident or very nearly so, and then further assumes that the stress corresponding to these two points is the least stress which if repeated will cause failure. The point is extremely interesting and it is to be regretted that the author did not quote experimental data to show the relation between the limit of repeated stress and the other properties of the material.

The author's suggestion that a steel of poor quality may have a normal value for the yield point and tensile strength (see Fig. 2) and an abnormally low elastic limit, and that this will correspond to a low resistance to repeated stress, may account for a number of failures of axles in which the tensile tests show normal physical properties. If so and if a determination of the proportional limit would have eliminated the material before it was put into service, this determination will merit most earnest attention. It appears, however, that considerable further research is necessary before any definite conclusion as to the quality of a steel can be drawn from its proportional limit. Mr. Mitinsky's paper is extremely interesting and if it stimulates thought and work along the lines indicated, it may prove of considerable practical benefit.

H. V. WILLE, Philadelphia, Pa.—The paper discusses the difference between the proportional limit, or elastic limit, and the yield point, and refers to the fact that much harm results from confusion of the terms yield point and elastic limit. Similar confusion has arisen in this country; the American Society for Testing Materials does not even consider the elastic limit and proportional limit to be identical and recently adopted the following definitions:

*Received June 3, 1915.

Elastic limit is the least load per square inch which produces a permanent set as indicated by an extensometer with a degree of precision of 0.0001 in.

Proportional limit is the load per square inch where stress (load per square inch) and deformation (elongation) per unit of length cease to be proportional to each other, the deformation being determined by an extensometer with a degree of precision of 0.0001 in.

Yield point is the load per square inch at which a distinctly visible increase occurs in the distance between the gage marks on the test specimen, as observed by using accurate dividers or an equivalent instrument, or at which, when the load is increased at the rate hereafter specified, there is a distinct drop of the beam on the testing machine.

It is customary in this country to determine the yield point for the softer grades of material, but the specifications distinctly require that the yield point is to be determined so that there can be no error resulting from the incorrect interpretation of the results.

Specifications for heat-treated material usually require a determination of the elastic limit and so that there can be no misunderstanding it is required to determine the elastic limit by means of extensometers. The determination of the elastic limit, of course, requires the removal of the load after every measurement is taken, in order to determine any permanent set in the test piece. The method followed is essentially the same as employed in the German laboratory. The proportional limit, however, can be determined without removal of the load, and can, therefore, be more readily determined than the elastic limit.

I have often observed a point of deflection in the stress-strain diagram which would indicate that the material had a very low proportional limit, but we have usually found this point of deflection to be the result of some inaccuracy in the measurements made by the extensometer as a result of one side of the test piece stretching more than the other side. None of our experiments show that it is possible to secure an abnormally low proportional limit in a bar which has been annealed at a rather high temperature or in one which has been finished at a high temperature. If the material has been manufactured from a homogeneous ingot our results show that the proportional limit is never less than the values obtained upon a thoroughly annealed sample of material. These results seem to be confirmed by those of Mr. Bauchinger who maintains that the number of reversals of reversed stresses of a given grade of material depends upon its homogeneity. Our own experiments on tires bear out this statement.

Fig. 1 shows a portion of a tire which developed a shell spot in service.

Fig. 2 shows a radial section of this tire which has been heated and quenched in water to develop any cracks which existed in the section of the tire. It will be noted that the tire contains a number of cracks

running at an angle of 45° through the periphery of the tire. These cracks are precisely of the same nature as those developed in testing a cylinder in compression. The cracks developed while testing such a



FIG. 1.—PORTION OF TIRE WHICH HAS DEVELOPED A SHELL SPOT IN SERVICE.

cylinder are shown in Fig. 3 and their similarity to the cracks in the tire will at once be noted. The results of this investigation showed that these failures resulted from two causes:



FIG. 2.—RADIAL SECTION OF TIRE, SHOWN IN FIG. 1, HEATED AND QUENCHED IN WATER.

First, lack of uniformity in the ingot from which the tire was made. Second, the presence of coarse grains or large crystals in the tire.

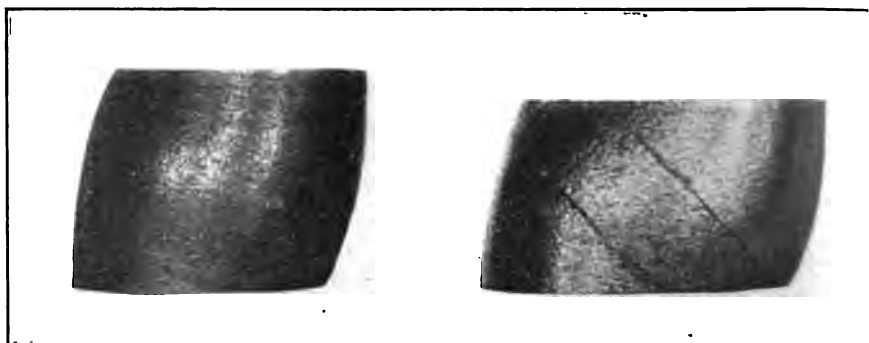


FIG. 3.—TEST CYLINDER SHOWING CRACKS DEVELOPED BY COMPRESSION.

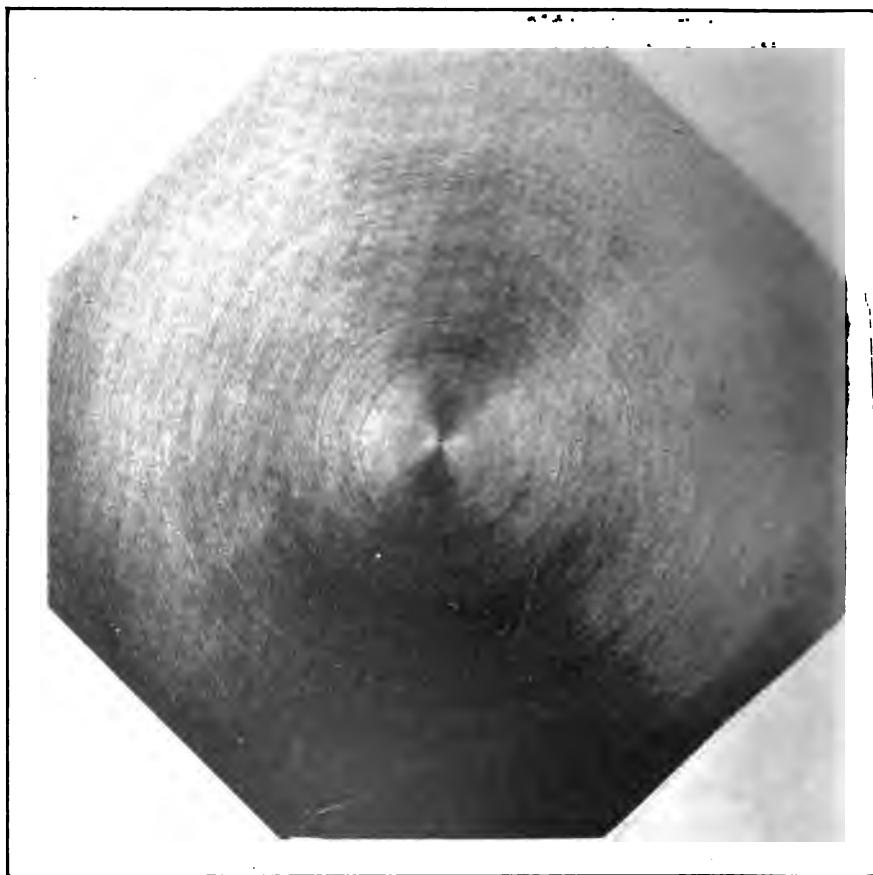


FIG. 4.—SECTION OF INGOT FROM WHICH TIRES ARE NOW MADE.

When a sample containing large crystals suspended in a matrix is subjected to a compression test the large crystals are pressed out of the matrix and when subjected to repeated loads either shell out or develop cracks.

Figs. 4 and 5 show sections of the ingot from which tires are now manufactured and their almost absolute uniformity will be noted from these photographs.

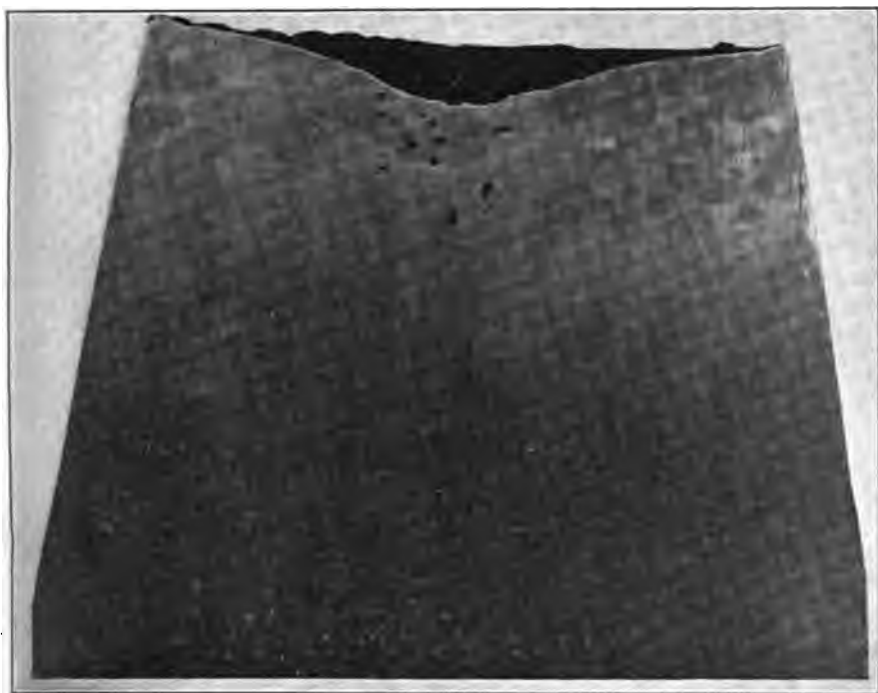


FIG. 5.—ANOTHER INGOT SECTION SHOWING UNIFORMITY.

Generally speaking, all results which we have obtained on fatigue tests show that the material of high elastic limit withstand a greater number of rotations under a given fiber stress than material having a lower elastic limit, irrespective of elongation or contraction of area.

The United States Government made a series of extensive experiments on this subject at the Watertown Arsenal on endurance of rotating shafts, the results of which are given in the following statement:

These results show an enormous increase in endurance of material having high elastic limits and generally confirm the views of Mr. Mitinsky.

I am also giving a statement showing results of endurance tests on steels of different carbon limit loaded with varying fiber stresses and these results likewise show the great endurance of the high-carbon steel or a

steel having a high elastic limit, irrespective of elongation or reduction of area.

Carbon	Nickel	Treatment	Fiber Stress	Rotations	Elastic Limit	Tensile Strength	Elongation	Contraction
0.26	3,282	Annealed	40,000	1,847,500	51,500	81,370	28.75	61.09
0.26	3,282	O., Q. & A.	40,000	1,815,200	66,950	90,640	26.90	65.04
0.25	4,514	Annealed	40,000	2,366,000	61,610	95,490	24.80	57.13
0.25	4,514	O., Q. & A.	40,000	3,296,700	101,860	120,190	20.80	60.05
0.29	5,661	Annealed	40,000	4,388,400	80,090	108,840	22.50	58.71
0.29	5,661	O., Q. & A.	40,000	3,795,200	117,610	131,410	19.65	58.88
0.539	27,353	Annealed	40,000	2,495,600	48,060	104,820	47.50	63.10
0.539	27,353	O., Q. & A.	40,000	1,088,200	48,850	97,780	43.35	60.80
0.24		Annealed	40,000	229,300	40,560	71,240	32.30	59.81
0.24		O., Q. & A.	40,000	348,000	45,170	74,440	33.15	69.93
0.42		Annealed	40,000	225,900	44,290	80,885	23.00	56.70
0.42		O., Q. & A.	40,000	655,600	55,000	92,180	26.05	57.22
0.46		Annealed	40,000	976,600	48,060	94,600	21.15	47.65
0.46		O., Q. & A.	40,000	1,657,500	61,110	102,880	23.05	51.27
0.66		Annealed	40,000	3,689,000	65,205	124,200	7.15	17.28
0.66		O., Q. & A.	40,000	4,323,600	92,040	154,920	13.50	31.48
1.094			30,000	50,000,000				
0.733			30,000	12,547,600				
0.824			30,000	16,336,200				
0.824			35,000	13,871,000				
0.094			35,000	19,152,300				

Tensile Tests and Repeated Stress Tests on Different Carbon Steels

Description (Steel)	Elastic Limit per Sq. In., Lb.	Tensile St'gth per Sq. In., Lb.	Elongation in 4 In., Per Cent.	Contraction of Area, Per Cent.	Mech'l Work at Rupture per Cu. In., Ft.-lb.	Max. Fiber Stress per Sq. In., Lb.	No. of Rotations at Rupture	Mech'l Work at Rupture per Cu. In., Ft.-lb.
0.17 Carbon	51,000	68,000	33.5	51.9	982	60,000	6,470	32,835
						50,000	17,700	62,635
						45,000	70,400	201,960
						40,000	293,500	665,290
						35,000	5,757,920	9,992,390
0.55 Carbon	57,000	106,100	16.2	18.7	1,047	30,000	*23,600,000	*29,500,000
						60,000	12,490	63,357
						50,000	93,160	328,000
						45,000	166,240	476,900
						40,000	455,380	1,032,130
0.82 Carbon	63,000	142,250	8.5	6.5	888	35,000	900,720	1,563,125
						30,000	*19,870,000	*24,838,000
						60,000	37,250	166,044
						55,000	93,790	399,780
						50,000	213,150	750,465
						45,000	605,460	1,736,910
						40,000	*17,560,000	*40,973,000
						35,000	*19,220,000	*33,635,000

* Not ruptured.

In this connection I would say that a method of testing has recently been suggested to determine the endurance value of a sample of steel without consuming a vast amount of time which this test requires.

It has been observed that when a test piece is pulled in a testing machine the temperature thereof falls as a result of the stretching or expansion in the molecules of the sample but when the molecules slide over one another or when a permanent deformation takes place, the temperature of the test piece rises. The load at which this change takes place has been very accurately measured and it is claimed that this value is a measure of the limit of indefinite endurance to repeated stresses. This is no doubt the same value as that referred to by Mr. Mitinsky and it is very much lower than the point at which a permanent set is first observable.

Our experiments likewise show that the size of the crystals rather than the tensile strength is a dominating factor on the anti-abrasive properties of steel. If the crystals are large it is evident that they are easily worn or broken while if they are small it will require a greater time to produce a given amount of wear. Our experiments further show that steel with a large-sized crystal will flow under load more readily than a steel with small-sized crystals and it is evident that the greater number of smaller crystals exhibit a greater resistance than a small number of large crystals.

These results are generally confirmatory of Mr. Mitinsky's experiments.

HENRY M. HOWE, New York, N. Y. (communication to the Secretary*).—Chief Engineer Mitinsky has put us under obligation by his extremely interesting paper on this very important subject. The total expense to which the community is put by our feeling obliged to make several different reception tests is very large, and could be greatly reduced if we could be confident that a single test were sufficient. The matter is so important that it is to be hoped that Mr. Mitinsky will favor us with a much fuller indication of his authority for his statements concerning the importance of the proportional limit.

Without wishing to deny the eight suggestions with which he closes his paper, I must say that there is much evidence which does not agree closely with what he says. For instance, in the extremely trustworthy work of Ewing and Humfrey¹ on very uniform Swedish iron with a proportional limit of 12 to 13 tons, they find that slip bands appeared after repetitional stress under a fiber stress of 6.9 tons, or not much over one-half of the proportional limit. When it was 7.3 tons, slip bands appeared and broadened in such a way as to suggest that they would in time cause

* Received Aug. 26, 1915.

¹ The Fracture of Metals under Repeated Alternations of Stress, *Philosophical Transactions of the Royal Society, A*, vol. cc, p. 241 (1903).

rupture, though the stress was only 58 per cent. of the proportional limit. Finally when the stress was 9 tons or more it invariably led to rupture, although it was not more than 75 per cent. of the proportional limit.

Other evidence showing that the proportional limit is materially beyond the safe working load for repeated stresses is the assertion of no less eminent authority than J. E. Howard.² He says that "a load of one-half the tensile elastic limit and less than one-third its tensile strength will in comparatively short time, by means of repeated stresses, rupture one of these rotating shafts."

My own interpretation, which I believe is also that of Mr. Howard is that the results which he had in mind in writing this were based on material in which the proportional limit had been raised unduly by cold rolling. Mr. Howard's expression "elastic limit" is to be interpreted as referring to the proportional limit.

The proportional limit is very greatly influenced by the finishing temperature and by cold rolling. It is, in my opinion, probable that the elevation caused by cold rolling within ordinary industrial practice is wholly fictitious as regards the endurance. So that in this respect the proportional limit alone is not a trustworthy guide to the endurance.

Mr. Mitinsky's eighth suggestion would, I think, be a closer approximation to the truth if it said that if the total range of stress from the greatest plus to the greatest minus stress does not exceed the proportional limit as determined in a tensile test, no number of repeated stresses will destroy the metal. That is to say, it is rather the total range of stress than the maximum stress reached in either the tensile or compressive phase that counts in tending to break the metal down.

I have yet to see any clear proof that the endurance is more closely related to the proportional limit than it is to either the yield point or the tensile strength. I do not deny that the proportional limit is the most trustworthy of the three, but I do say that I have not yet seen evidence to prove that it is, and that I should value evidence to this effect very greatly.

His statement "the proportional limit cannot however be carried above the yield point"³ seems to require some explanation. Will he favor us with his authority for this assertion?

While it is true that the artificial raising of the proportional limit in tension by repeated stresses is accompanied by corresponding reduction in the proportional limit in compression, this is not true according to my own observations of single applications of tension and compression. On the contrary, I find that a single application of tensile overstrain is

² *Trans.*, xxiv, p. 777 (1894).

³ *Bulletin* No. 104, p. 1699, (August, 1915.)

sotropic in its action, strengthening the metal in compression and tension equally.⁴

MANSFIELD MERRIMAN, New York, N. Y. (communication to the Secretary*).—The conclusions of the author regarding the value of the proportional limit seem valid for static stresses and also for stresses due to loads applied without sudden shocks. For bridge members I agree that the proportional limit is the most important property to be specified or determined by test. Perhaps a sudden shock may be defined as an application of a load which causes local injury to the material. Railroad rails are subject to such shocks due to imperfections of track and running gear; moreover the rail itself may have defects due to lack of care in manufacture and these are especially liable to develop further under the action of sudden shocks.

The drop test now used in this country aims to imitate the shocks that occur to the rail in service and also to discover the internal defects of the metal. This test has proved very valuable in insuring better qualities of rails than would otherwise be possible.

I am of opinion, however, that a test for proportional limit ought to be introduced into rail specifications. The arguments of the author in favor of this seem sound except that he does not sufficiently consider sudden shocks. For resistance against such shocks the drop test seems necessary. But under fair conditions of track it is found that the percentage of rail failures is very small, and hence it appears that the quality of the large percentage of rails which do not break might be improved by introducing into the specifications a requirement for proportional limit.

JAMES E. HOWARD, Washington, D. C. (communication to the Secretary†).—The remarks of Mr. Mitinsky are directed to methods of test having to do with railway material. The special feature, which is peculiar to the Russian specifications for rails, consists in the determination of the elastic limit of the finished shape, inferentially making that value the basis for the working stresses in the track.

The author remarks upon the position held by Hooke's law in the calculations for engineering structures, without which it would be impossible to design such structures intelligently; refers to the elastic limit as the important factor in establishing a limit on the allowable

⁴ Are the Effects of Simple Overstrain Monotropic? *Proceedings of the American Society for Testing Materials*, vol. xiv, 1914. *Étude sur les Propriétés Physiques et Mécaniques des Métaux, Congrès International des Méthodes d'Essai, Paris Exposition*, vol. i, p. 270 (1900). See also Galy-Aché, *Mémoires, Revue de Métallurgie*, vol. x, p. 587 (1913).

* Received Sept. 7, 1915.

† Received Oct. 1, 1915.

stresses, and the need of fixing the limits of the working stresses with reference to the elastic limit.

Those tests are desirable which furnish exact information on the physical properties of the materials which are used, hence the determination of the elastic limit of the rail commends itself. The acquisition of this information also carries with it the associated idea that equally exact knowledge will be acquired concerning the stresses in the track, in order that the working or track stresses shall bear their proper relation to the elastic limit of the steel. The argument for exact testing loses much of its force if the stresses to which the materials are to be exposed remain undefined.

As a method of test there appears no practical difficulty in determining the elastic limit of the rail. With the introduction of such testing facilities as would naturally follow, the determination of the elastic limit might be done very expeditiously, closely competing with current tests in respect to the total time of handling the material. Tests, however, are not to be judged according to the time required to make them, but fundamentally from the information which they furnish.

The drop test perhaps has the distinction of furnishing about as limited an amount of information as any which may be mentioned. It has not been claimed for it that it tells the value of the elastic limit. It is not uncommonly supposed to have a relation to service conditions of the rail in the track, although the relation is not as close as at first sight appears.

The striking velocity of the tup of the drop test is much lower than the velocity of high-speed trains. The blow of the tup is one of retarded velocity and, unless the rail fractures, comes to a state of rest at the time of maximum strain. The loading of the rail, as accomplished in the track, is without retardation of speed. So far as known, there is no lag in the display of elastic extensibility in steels by rapidity of loading. From this it would appear that the part of the drop test having to do with the elastic extension of the metal was without special efficacy. The final portion of the test, during which the rail receives a permanent set, constitutes a stage which does not or should not have its counterpart in the track.

It is desirable, however, that information be at hand on the effect of rapidly applied loads, of speeds up to and beyond ordinary train speeds, with loads capable of giving different amounts of permanent sets. Such tests possess a research character, upon which the ordinary drop test has very little bearing. There was occasion in a report of the Interstate Commerce Commission, dealing with a broken rail which caused a derailment in December, 1911, to refer to a test made under static conditions in a testing machine, saying: "This test permitted of the determination of the elastic limit of the rail, an important factor in the

strength of materials, but not ascertained in the prescribed drop test." On that occasion it was also shown that the elongation in the drop test was from 5 to 7 per cent., while in the testing machine the rail bent through an angle of 30° without rupture, the extension of the metal reaching 16 per cent. It is the function of a research inquiry to develop the influence of rapidly applied loads taken at different temperatures and the effect of cross-section of rail. It is not established that a limited range in striking velocities, varied according to the weight of the rail, has real significance and furnishes reliable data on the properties of the rail.

These thoughts lead to giving a preference to a test which has the merit of furnishing a well-defined value, and one which in apparently all engineering matters, except rails, is recognized as the factor most needed.

Concerning the sharpness of definition of the elastic limit, that will depend upon the grade of the steel and the rate of cooling and relative thickness of the different parts of the rail. Internal strains in cooling and differences in the elastic limit of the thick and thin parts of the cross-section will ordinarily result in some vagueness in establishing the elastic limit of the rail as a whole. Failure to recognize influences which tend to cause local sets has led to confusion in judging of the elastic limit of steels. It is well to bear in mind that it has not been shown that any change in the value of the modulus of elasticity occurs until there is an overstraining load on the metal.

Methods of test form only a part, and perhaps a minor part, of the rail question. In fact, it almost seems futile to consider the careful determination of a property of the rail when service conditions so generally result in overloading that particular feature to which attention is directed. Rails present an example in which, very commonly, there is an overload, exceeding the elastic limit, the vital question touching upon the ultimate result and what constitutes the elements of safety in the use of the rails under such circumstances.

The cold rolling by the wheels, when the rail is in the track, introduces internal strains which are evidence that the elastic limit of the steel has been surpassed. This is a matter of common occurrence from which few if any rails are exempt. The higher the wheel loads, the deeper the penetration of their effects, and the more of the metal thus disturbed.

The internal strains introduced attain values not infrequently exceeding the direct bending stresses of certain of the wheel loads. They are strains of compression in the metal immediately next the running surface of the head of the rail, while there are strains of tension in the center of the head which furnish the necessary reaction to the compressive strains. In some recent tests conducted by the Interstate Commerce Commission, in connection with rail failures, compressive strains were measured in rails from service exceeding 21,000 lb. per square inch

equivalent stress and over 9,000 lb. per square inch tension in the center of the head. It will be seen that these results explain the formation of the development of transverse fissures in the heads of rails.

These internal strains which are set up by the action of the wheels on the head, on the running surface and on the gage side of the head, do not appear to have attracted attention, although they seem to afford a direct measure of the effects of high wheel loads. In combination with the bending stresses, they reduce the magnitude of the tensile strains in the fibers of the head most remote from the neutral axis of the section, while increasing the tensile strains of the interior fibers. The cold rolling embrittles the steel, as made manifest in the loss of ductility in rails from service when loaded to destruction, with the head in tension.

In a practical way, the greatest interest attaches to such tests as are calculated to insure the acceptance of rails which give the best service in the track. Chemically hard rails resist abrasion in a very satisfactory manner. They have furnished, however, a large percentage of those rails which have displayed transverse fissures. Tests for acceptance do not cover such cases of failure. Against fatigue failure, by repeated stresses, current tests furnish no data. It would be an aid to have the elastic limit of the rail determined, still the influence of the internal strains from the action of the wheels introduces a factor concerning which there is lack of information.

The development of transverse fissures seems to be the only type of failure due to repeated stresses, the type chiefly or solely due to stresses which approach the elastic limit, in longitudinal tension. To prolong the life of the rail, one or both of the component factors should be reduced, either the section of the rail so increased that the longitudinal tension shall be materially decreased, or the wheel loads reduced to effect a reduction of the internal strains.

In discussing the rail question it is rather difficult to keep close to the matter of specifications and tests. Features not directly pertaining to tests and specifications, as they are being currently carried out, force themselves into prominence and make the tests appear subordinate to other considerations.

On the one hand, high wheel loads and hard steels show a tendency to develop transverse fissures, constituting a track condition; on the other hand, the presence of laminated and seamy steel leading to split heads and base fractures constitutes a mill condition. Those are the vital features which have to do with a large share of rail failures.

It would appear to some that knowledge of the chemical composition of the steel and the finishing temperature of the rail would furnish information from which the elastic limit of the rail might be judged. That information would probably constitute the basis on which the steel maker would proceed in the manufacture of the rails to meet given

specifications for the elastic limit. This idea carried out might lead to less testing and more complete knowledge of the processes of manufacture. The fact of making a test implies there is a doubt concerning the product. The test piece does not remove the doubt concerning the next heat of steel, nor furnish absolute data concerning other portions of the heat it is supposed to represent. The principal safeguard lies in the possession of knowledge of the processes of manufacture, in the production of the desired grade and quality of steel; then, since there is an ultimate limit to the endurance of all steels, adopting and providing such conditions of service as the steel is capable of sustaining with safety.

GEORGE K. BURGESS, Washington, D. C. (communication to the Secretary*).—Mr. Mitinsky's paper, embodying as it does descriptions of test methods so greatly at variance with those practices with which we are familiar in specifications for railway materials, is of very great interest and value. That it is a summary of some of the methods employed and results obtained by the Department of Tests of the Russian State Railways, gives very great weight to the conclusions.

To define and interpret adequate tests for some of the classes of railway material, such as axles, tires, rails, and wheels, has been and still is a very perplexing matter; and the types of specification for these materials are being constantly questioned and revised.

Mr. Mitinsky concludes "that if a rail or a tire shows a high proportional limit and sustains satisfactorily a drop test, the steel is of as good quality as can be desired;" all other tests, both chemical and physical, he considers superfluous.

This, of course, is in strong contrast with many American specifications for articles of steel, with their rather sharply defined chemical limitations, oftentimes accompanied by manufacturing restrictions and complemented by physical tests of various kinds.

Most of the European countries also have even more elaborate acceptance tests of railway materials such as rails and tires; although, with the exception of the United Kingdom, where for rails, the limits of C, Mn, Si, P, and S are specified, as is the United States practice, the chemical requirements are, in appearance at least, less exacting for rails. Thus, in Italy only the maximum P and S contents are specified, in France C and P or C, or even nothing chemical is specified depending on the railroad, and this last is also the practice of Holland, Austria and Germany, while Sweden specifies only the P content.

There is as great variation among the physical tests of rails required in the several countries, and in the United States, from one railroad to another, in spite of the standardizing societies. There is the practically

* Received Oct. 23, 1915.

material in service, tested by the methods he describes, would be of great interest.

The following incident is not without significance: Not long ago, I was showing to a Russian engineer of railways a typical case of the failure known as "transverse fissure" in a rail; his comment was "That could not occur in Russia."

GAETANO LANZA, Philadelphia, Pa. (communication to the Secretary*).—In this paper Mr. Mitinsky makes a strong plea for determining the proportional limit in practical commercial testing, and for basing upon it the value of the working strength to be employed in designing. He also deplores the too general, erroneous custom of confusing it with the yield point, and seeking the latter instead of the former, in order to save time in testing. He points out that the two are very different; that from the one the other cannot be inferred, and that the yield point is not a suitable quantity upon which to base the working strength to be used in practice. He calls attention to the fact that, if we employ a continuous motion of the machine, a slow speed, and apparatus of sufficient accuracy and delicacy, the experimental determination of the proportional limit need not take more than 15 or 20 min.

As to basing upon the proportional limit the value of working strength to be adopted, I will say that in my opinion its determination ought to rest upon a consideration of the proportional limit and the ultimate strength together, and not upon one of them to the exclusion of the other. Indeed, were the ultimate strength disregarded, it would be possible to so treat the steel as to raise the proportional limit artificially to such an extent that the working strength chosen would be too large a fraction of the ultimate strength to insure safety.

The paper seems to imply that the proportional limit is identical with the breaking strength for repeated and alternate stress. This is probably approximately true for perfectly sound material free from sharp corners and from abrupt changes of section, but in the case of such metal as is in common use, with its ordinary defects, blow-holes, segregation, etc., and with more or less sudden changes of section, the proposition is at least doubtful, and much experimental work on repeated and alternate stresses is needed to determine the actual facts. In the light of our present knowledge it would not be safe to assume that the results of such tests could be predicted without making them.

* Received Oct. 25, 1915.

The Pacific Coast Iron Situation

Discussion of the paper of CHARLES COLCOCK JONES, presented at the San Francisco meeting, September, 1915, and printed in *Bulletin* No. 105, September, 1915, pp. 1887 to 1898.

D. A. LYON, Salt Lake City, Utah.—For some years I have been interested in the iron-ore deposits of the Pacific Coast, and I feel very much indebted to Mr. Jones for the information he has given us on this subject. I think that we are also very much indebted to him for having presented this paper, as he has given us reliable data in shape of facts and figures, which is just what we need in considering whether or not it would be commercially feasible to attempt to establish an iron and steel industry on the Pacific Coast. Personally, due to lack of market, I do not think it is a commercially feasible proposition at the present time. Mr. Jones states that pig iron can be produced on this coast as cheaply as it can be produced in Pittsburgh. Even so, what can be done with it after it is produced? To begin with, iron produced on the Pacific Coast has to compete with iron produced in China. Last year I had occasion to investigate the subject of the market for iron and steel on the Pacific slope, and found that it was possible for the producers of Chinese iron to sell it for much less than the price for which iron can be produced in the West, and also for less than eastern pig iron is now sold for, even when shipped through the Panama Canal. I was informed that the agents for Chinese iron had instructions to sell it for \$1 or \$2 less per ton than that asked for other irons on the market. Of course, I realize that the competition of Chinese iron may not be important, but even if it is not, and even though pig iron can be produced at the cost given by Mr. Jones in his paper, I do not see any immediate chance of its being so produced, due to lack of market.

C. C. JONES, Los Angeles, Cal.—Mr. Lyon is right. It is a fact that Chinese pig is being sold in San Francisco on some large contracts at \$18.50 a ton, while \$19 is the usual price. We ought to produce iron on this coast at \$14. Personally I do not think Chinese iron is ever going to be an important factor. It is a bugbear out here now. You are not going to have much Chinese pig iron in this country. Our little friend Japan is going to absorb everything it can. The price of Chinese pig iron at \$18 is about as low as it will ever get. If we can show you that we can produce this iron here at the same price as at Pittsburgh, I think you will have a good opportunity to compete with anything China or India can ever do. Costs there are very low now. They are going to come up by leaps and bounds in the next few years. The labor cost over there may be very low now, but I have seen some men from there quite recently

who say the costs are going up. Those people can not live under the conditions that are imposed upon them at the present time.

The best market is not in rails and heavy structural steel but in the smaller diversified industries. That is the point I am trying to make. The idea is not to try to take away the steel rail business or the big steel plate business from Pittsburgh but to install a blast furnace and equipment to make special steel.

CARL H. BOOTH, Chicago, Ill.—Outside of a lack of a market I feel that there is one other weakness in the situation at this time, and that is the lack of men who know how to do the work. In my rather brief investigation, I find that there are few men on the Pacific Coast really experienced in steel work. With the exception of charcoal and coke, the natural resources are plentiful and the materials required for the manufacture of iron and steel can be obtained at low cost. My chief criticism has been this lack of men with proper experience. I have just come from a trip to Los Angeles. There are a number of machinery manufacturers there who freely complained about the quality of the steel castings produced on the Coast. Most of them are buying their best castings in the East simply because they cannot depend on the quality of the local product. There is no question in my mind that if experienced steel foundry men are induced to come out here, the iron and steel business can be started properly and successfully. It stands to reason that if the same brains and intelligent handling of the proposition were given to it here as in the East, there is no reason why the resources here should not be developed, and there is no excuse for the Pacific Coast buying its castings back East, when the same application and knowledge could be applied here and castings produced that would be just as good. I am talking now about the manufacture of the product after the steel is made, but there has seemed to be, from the investigation I have been able to make in the last three or four weeks, a lamentable absence of experienced handling. I believe that there is already a large market on this coast for steel castings. If you have both the natural resources and the market here, you must also have the ability to take hold of it and deliver the proper product.

BRADLEY STOUGHTON, New York, N. Y.—It would seem to me that the discussion of this paper was departing from the established principles of political economy. Mr. Jones has given us in his paper some very carefully culled figures and the question should be, not whether somebody else could supply the material listed by Mr. Jones at a price less than is now being paid, but whether the figures themselves are correct.

The question as to whether China can or cannot send iron to this country at a lower price than is now being paid for it, and the question as to whether the iron foundries of California do or do not employ suffi-

ciently skilled men to make good castings, could be discussed from many different viewpoints, but if the figures presented by Mr. Jones are correct, and if, as stated by him, the greater amount of iron and steel used on the Pacific Coast is now obtained from the East, then there can be no question that there is a real market here.

Looking at the matter from another standpoint: It would appear that the Pacific Coast with the greatly enlarged scope for industrial activity that is given by the operation of the Panama Canal, including manufacturing of many kinds, exporting, and shipbuilding, must use in the future greatly increased amounts of iron and steel products.

D. A. LYON.—I think Mr. Stoughton has laid too much emphasis on what I said in regard to the competition of Chinese iron. Although we do have that competition at present, it may not, as pointed out by Mr. Jones, affect the pig-iron and steel situation in the future here on the Pacific Coast. We will assume that it will not; but even so, and even if iron can at present be produced as cheaply at certain points here in the West as it can be obtained from Pittsburgh, now that the Panama Canal is open, will we not be able to get iron and steel from Pittsburgh cheaper than we do now?

C. C. JONES.—You would have to pay the freight rate from Pittsburgh to New York, then you would have a freight rate of \$4.50 or \$6 from New York to San Francisco or Los Angeles. When you add that it makes very little difference.

D. A. LYON.—Even if we could produce it here as cheaply as they can in Pittsburgh, I still maintain that we do not have the market. We have with us Mr. Lash, one of the most prominent iron and steel men in the country, and I am sure we would be glad to hear from him on this subject.

H. W. LASH, Cleveland, O.—I think, as Mr. Lyon has pointed out, one of the troubles is a market out here. There is a large consumption of iron and steel on the Pacific Coast, but it includes nearly everything that is produced in the line of iron and steel and there is no one plant that could cover all these different grades and shapes without a tremendous outlay. That is one thing that would interfere more than any other with the production of steel in large quantities on this coast.

As for pig iron, I think there is a greater opportunity. There are many foundries in California that have to bring their iron from the East or from England, and there has been some iron sent here from India and China. I think there is a chance to develop the pig-iron industry, though probably not to any very great extent. There is surely quite a consumption here, and a small plant properly located and properly managed

ought to be able to compete with the East if the raw materials exist in the quantities represented, and they no doubt do.

J. W. RICHARDS, South Bethlehem, Pa.—I would like to ask Mr. Lash if he will give us his opinion about the steel-casting industry on this coast?

H. W. LASH.—I have had occasion to visit one or two steel-casting plants, one at Pittsburgh, Cal. They have a very good plant and are turning out good product. They say they have all the business they can possibly turn out. Their castings are made from open-hearth steel and while they have not attempted the smaller castings similar to those made in the East, they will no doubt take up that class of work soon. In my opinion the casting business has quite a future here. The matter of delivery is very much in their favor. Customers do not care to wait for the long time of delivery from the East.

C. C. JONES.—I understand that there is an electric steel foundry near Portland which is in operation. It strikes me that the situation here is particularly favorable for the introduction of an electric furnace to make steel castings from scrap, because there is a very large quantity of scrap on hand which can be melted and recast into shapes. I personally believe that right here in San Francisco or Oakland, right on the San Francisco Bay, an electric steel-casting plant could be quite successfully operated.

D. A. LYON.—I sincerely hope that I will not be considered a pessimist as regards the possibility of producing iron and steel on this coast. In this discussion I am thinking only of large tonnages. What Mr. Jones says is true. There is a chance out here to develop an iron and steel industry which will supply the iron and steel that is needed for local consumption. However, there is not a market at the present time for large quantities. It is true that the sum total of all the various kinds of iron and steel used amounts to a considerable tonnage. A great deal of that, however, is needed to supply parts of machines or implements which are manufactured in other districts of the United States. Then, too, the variety of shapes needed is too great for any one plant to supply. If, therefore, the local demand is to be furnished by local plants, it seems to me that it must be done by small plants. In other words, I do not believe there is a demand at the present time for a large iron or steel plant on the Pacific Coast, such as we find around Pittsburgh.

C. H. BOOTH.—I believe there are a lot of concerns out here waiting for an opportunity to use steel in place of iron castings. I have made a somewhat detailed study of California conditions. There are a great

many companies that would use steel castings if they could get prompt delivery. This is especially true of small steel castings. The heavy castings are made by the plant you spoke of, Mr. Lash, but the smaller castings today are made with converters or crucibles with the exception of one or two electric furnace plants and the quality has not been altogether what could be desired. Outside of delivery and quality, I think other things are mere matters of detail, yet they are important. A great deal of the heavier iron castings out here will be replaced by lighter castings made of steel, if they can be obtained at a reasonable cost. It looks to me as if the opportunity is a very large one, because you have electric power here at reasonable rates, and steel scrap today in this market costs from \$5 to \$7 a ton. You cannot buy scrap at any such price in the East.

MR. MALM.—I notice that Mr. Jones mentions a deposit in Shasta County and I would be glad to have any information regarding what they did up there with their electric furnace.

J. W. RICHARDS.—That company started to make pig iron in an electric furnace and discontinued it, and has recently started up a furnace to make ferromanganese and ferroalloys, and they are now making them very profitably.

MR. MALM.—They discontinued the manufacture of pig iron up there.

MR. RANDALL.—I was told it was due to the high cost of coke. They had some intention of trying to make it in a charcoal furnace and the Forestry Service was trying to sell them some of their wood.

MR. WILLESTON.—The reason for their shutting down was due to the fact that they could not get charcoal at a low enough price. They installed a large coke plant, but it proved a failure. They could not produce charcoal economically, and they could not buy it at a reasonable figure, so it put them out of business.

C. H. BOOTH.—I have just returned from a visit to that plant. I spent a day there about a week ago. I had an opportunity to talk with one of the owners of the plant, and he said he did not believe they would ever attempt to make pig iron again, that it was not commercially practical. They made some very good pig iron but they could not compete with the cost of other pig iron brought in.

C. C. JONES.—These remarks have helped this subject. Mr. Lash is correct in saying the market here is largely one for pig iron. My idea has been that the establishment of a single furnace of 250 tons capacity

would lead to so many other things that would not compete with the larger tonnage, that in the end we would have half a dozen little industries that would mean more in wages and in selling profits than tonnage represents. Take the matter of cast-iron pipe alone. In Los Angeles County today there are contracts aggregating \$2,000,000 for this cast-iron water pipe. People are going into the business there with small experience, using poor pig iron with a hope of filling some of these contracts. That is Los Angeles County alone, and does not include the contracts for cast-iron pipe that are being let around Stockton, Portland, Seattle, and other places. We have a pretty long coast line here, nearly 2,000 miles, that is going to fill up mighty rapidly in the next 20 years. Anything that is based on an increase of population ought to be a success commercially. As Mr. Lyon has said, it is a commercial matter.

BRADLEY STOUGHTON.—We have heard it said in the East that the difficulty of establishing an iron and steel industry, or any industry, on the Pacific Coast is one of elasticity of labor supply and I should like to ask if Mr. Jones has considered this question and what conclusion he has reached thereon.

C. C. JONES.—I cannot say that I have looked into the question of the elasticity of the supply, but I have heard it said that Eastern manufacturers would not go into manufacturing here on account of labor conditions. But it is not the supply. Labor is looking for a fight all the time; the union features are very strong here.

BRADLEY STOUGHTON.—Mr. Schwab has some very large interests on the coast, has he not?

C. C. JONES.—Yes, the Union Iron Works, for one, and he probably supplies large amounts of structural steel to firms for fabrication.

H. W. LASH.—I have had some little experience out here, during the last two years and have found the quality of labor very good. I believe the average equals that of Ohio and Pennsylvania, and the climatic conditions are surely more favorable. We have neither intense heat in the summer nor extreme cold in the winter. There is no doubt that labor unions are very strong. In some cases they are unreasonable, but I believe the labor situation here is somewhat exaggerated. I also find there is something about California that is rather inviting and that workmen are quite ready to come here if they can be furnished employment similar to that they have in the East. A great many like to come and try it, and when they do come they usually want to stay.

Electrometallurgical Industries as Possible Consumers of Electric Power

Discussion of the paper of DORSEY A. LYON and ROBERT M. KEENEY, presented at the San Francisco meeting, September, 1915, and printed in *Bulletin* No. 104, August, 1915, pp. 1707 to 1730.

LAWRENCE ADDICKS, Douglas, Ariz.—I think papers of this character, while general in nature, are of particular value and interest. because when we figure on competing with the East in electrometallurgical industries, on account of the cheap power on the Pacific Coast. we have to take two or three things into account. In the first place. power very rarely amounts to more than 20 per cent. of the total cost of an operation, even though it be electrometallurgical, and I think it is the common opinion among those who are not familiar with those particular industries, that it amounts to more than 50 or 60 per cent. Take copper refining; we may say that it is 15 per cent. of the total cost. In the second place, around New York City, with 100 per cent. load factor and a reasonably large load, say 10,000 kw., we can generate from a certain class of coal 1 kw.-hr. for about 0.3c. I might say, however, that 0.3c. does not include overhead charges, interest on the investment, etc. The third factor we have to consider out here is that any product to be salable must command a retail market. It would be useless to produce wire bars here on the Pacific Coast because there is no place to roll them. If we were going to establish a copper refinery, that part of its output which goes into wire bars would have to be taken care of by building a rolling mill alongside of it to turn it into wire and finally make a product that is salable. I do not think, however, that Mr. Lyon need have excluded copper refining from his industries. There are perhaps 50,000,000 lb. of copper a month refined in this country on a custom basis. I think the smelting companies would be very glad to build refineries here if conditions warranted it. The recent extensions of the Great Falls refinery and of the Tacoma refinery in the last few months give evidence of that.

CARL H. BOOTH, Chicago, Ill.—I agree in a general way with the features of Mr. Lyon's paper. I have read it with very great interest. It seems to me the real reason for the lack of development of electrometallurgical processes on the Pacific Coast is largely the lack of a market. I am not claiming to have a great knowledge of conditions out here, excepting those I have personally observed in the last few weeks, but my general conclusions are that in the one field of electric steel melting and refining, regardless of the low cost of melting, the lack of a proper market will always be a severe handicap to any operation of that kind on the

Pacific Coast. Even with cheap power and low-cost materials, the limited tonnage available is a distinct disadvantage. If the steel made here must go East to find a market, the cost of freight rates, and the fact that power can be produced almost as cheaply in the East, will more than offset any other advantage; so that my general thought is, being especially interested in the line of steel metallurgy, that the chief drawback to the manufacture of a heavy tonnage of steel on the coast is the lack of a market.

Apparently the authors of this paper are not entirely familiar with some of the recent developments in the electric steel melting field. Electric furnaces are today being built which produce steel cheaper than the fuel melting furnaces for the same tonnage. These facts have been well worked out and carefully checked for a period of two years. Such furnaces are today producing steel at \$18 a ton with current at $\frac{3}{4}$ c. per kilowatt-hour, and materials at \$11 per ton. The upkeep cost is low and the efficiency of the furnace is so high that it has been practical to obtain low-cost operations, and to use electric power and still compete with fuel melting. With this one point of difference, the cost of melting with the electric steel furnace, I thoroughly agree with Mr. Lyon's conclusions. I would call your attention especially to p. 20, where he gives a schedule of the operating cost of an electric steel furnace. With current at the price he gives, steel could be produced for about \$18 a ton, so that there is at least a difference of \$12 a ton in Mr. Lyon's figures.

It makes no difference how cheap you can make the steel out here; if you have no market for it, it is hardly possible to do much with the product. Even if steel is substituted for iron, with the increased tonnage that might be obtained still I can see no hope for developing a large industry here in the immediate future.

J. W. BECKMAN, San Francisco, Cal.—I take pleasure in disagreeing with the attitude that both Mr. Lyon and Mr. Booth assume in regard to possibilities on the Pacific Coast. I think it is undoubtedly true, if you only look to the near future, that in the main they are correct in saying that there are small markets only, but this is a new country, and we are practically in the same position here as the East was before the Civil War. Before the Civil War the East would not do any manufacturing because they thought they could not compete with England. As they could buy what they needed from England they saw no reason for manufacturing. The East has changed! It has been shipping for many years, and for a long time to come will ship manufactured products to England. Here on the Pacific Coast everybody apparently is looking to the East for their markets. I have often heard said: "What is the use of manufacturing out here, they can beat us in the East?" I think that is a wrong attitude for the people of the Pacific

Coast to take. We are favored with enormous quantities of water power here practically at tidewater. There are a number of opportunities right on tidewater, and a number just a few miles away from tidewater. We have, therefore, the same power facilities on the Pacific Coast to produce electrochemical materials that Sweden and Norway have, especially Norway, and the same opportunity for shipping these materials to the world's markets. My feeling is very strong that the future of the Pacific Coast does not lie east from here, but west and south. We have over half of the world's population facing the Pacific Coast. We should take care of the electrochemical needs of that population and we will do so, provided we look at matters in the right way. If we look only to the east we are certainly going to let everybody in the East, as well as Europe, take care of those markets that legitimately belong to us. Therefore, I feel that the broad-minded view is not to look forward to what this generation can do, but build for generations ahead and start doing so now. That is a duty resting on the present generation on the Pacific Coast. There is an enormous future for this coast. I think a great part of the people on this western coast do not appreciate the opportunity this section of the country affords. We have the Chinese and Japanese markets of unlimited possibilities; we have the northern part of Siberia, that we do not know so much about; there are also markets to be had in South America and Australia, and we might reach for markets even as far as India, and in years to come I believe my attitude will be proven correct. I think this attitude of belief in the possibilities of this coast and its future is psychologically the right one to hold. Mr. Lyon does not seem to have taken into consideration the fact that we can get aluminum oxide at a very low figure here, from a deposit in Utah that is being worked now, and great quantities of practically pure aluminum oxide are obtained from it. It is claimed that the product is obtained so pure that in many cases there is no need of further refining. A reasonable freight rate has already been established by the Western Pacific R. R. for this aluminum oxide from those deposits to the Pacific Coast. This aluminum oxide is obtained as a by-product at a very low price, and it seems as though there is a big possibility for making aluminum on the coast from this raw material.

Mr. Lyons says that all our electrodes are shipped here from the East. We have here on the Coast one very large and permanent source for the manufacture of electrodes in what is generally known as lampblack. In the making of volatile products from oil there is always produced quite a quantity of exceedingly fine carbon with some heavy oil mixed in. In many cases this has been used as fuel under boilers, because of no other use for it. From my knowledge of electrodes I believe that there is a possibility of using this practically pure carbon in the making of elec-

trodes, and using them in aluminum manufacture and in electric furnace work.

Pig-iron manufacture is a very old process and no radical change has taken place in its manufacture until comparatively recently. Is it not possible that such an extremely old process will be substituted in time by an absolutely new process utilizing new agents in it? Here on the Pacific Coast, for example, we have an abundant supply of natural gas and an abundance of fuel oil; we also have an abundance of cheap electric power. Couple with these an abundance of cheap iron and it seems as if there must blossom forth a large iron industry.

Along the coast there are great deposits of iron ore. About 200,000,-000 tons are now located, running 64 per cent. of iron. It is of interest to note that the United States Steel Corporation is running at the present time on 56 per cent. iron ore.

Why should we not look forward to the utilizing of new agents and methods in place of the old processes for the manufacture of pig iron?

Another point that has impressed me in connection with Mr. Lyon's paper, and to which I have not referred, is the question of treating refractory ores by electrolysis in fused electrolytes. I believe it might be possible by such a process to partly separate the different metals present in the ores.

LAWRENCE ADDICKS.—Mr. Beckman has put the problem on its true basis. If we did not have cheap power here, the whole matter would be impossible. Of course, in regard to aluminum, we all of us have an aluminum mine in our back yards, but the difficulty has always been to get a deposit of value. Will Mr. Seaver tell us what the refracting companies have done in the direction of using alumina?

KENNETH SEAVER, Pittsburgh, Pa.—I hardly feel that I am qualified to speak with any authority about what has been done by the refractory companies in this field. You are all familiar with a brick made from hydrous oxide of aluminum. I speak now, in the first place, of a content of about 56 per cent. in the brick, and another of about 77 per cent. of alumina. The characteristics of the two bricks vary rather widely. Those two are probably the extremes of alumina commercially valuable to the refractory industry. The 56 per cent. aluminum content is used for fire-clay brick, but it will withstand higher heat. We are here considering alumina as neutral, although we are familiar with its action under varying chemical conditions, and this 56 per cent. content aluminum brick has been used where reasonably high heat was encountered. The other material running as high as 77 per cent. in alumina content has thus far been used as a substitute for magnesia brick, or more often termed magnesite brick, in the open-hearth and similar processes. It has been giving excellent results, but those results in the light of years

of experience, should not be taken for more than they are worth. It is not yet safe, in my opinion, to say that it is a substitute. I believe that it is. Under normal conditions it probably cannot be produced at the price at which magnesia brick was produced. This particular refractory has been used in the port end of a furnace, forming a bulkhead of that one particular open-hearth furnace, for 365 heats, at which time the furnace was necessarily taken off to install a new roof. At that time the opposite port, which was installed with magnesia brick alone, was in a worse condition. The indications are that this 76 per cent. alumina brick will give equally good results. On the other hand, I feel, bearing in mind the experience of years, that a thing of this kind must be tested out under every conceivable condition before one can make a flat statement on the subject. I believe it shows great possibilities, however.

J. W. RICHARDS, South Bethlehem, Pa.—I wish to speak on several matters connected with the subject of this paper. The first is the possibility of producing aluminum on this coast. Mr. Lyon considers the cost of aluminum here and shipping it to New York. I believe that if such a plant were established here the material would not be shipped away, but would be used to supply the local market, and also the Japanese market which is a considerable one at the present time. The principal raw material is bauxite, or its equivalent, and the question is whether it can be found east of the Rocky Mountains. If the byproduct alumina from the Maryvale alunite deposits in Utah could be brought here at \$5 per ton, that might put a different aspect on the matter, or, if bauxite were to be found on the Pacific Coast there would be a possibility of establishing works here to supply the local market and the Japanese or other foreign markets bordering on the Pacific Ocean.

In regard to the consumption of steel on this coast, in the paper by C. C. Jones of Los Angeles, it is stated that the consumption of steel on the Pacific Coast is nearly 3,000,000 tons per year. That is not an insignificant market, although I got the impression from Mr. Lyon that he considered the local market here was not worth considering. That is a respectable proportion of the output of the United States. If there is such a market here, and the statements made as to the cost of making steel in a properly designed electric furnace are to be relied upon, there is a distinct possibility of the Pacific Coast furnishing its own supply of steel castings and possibly some manufactured shapes. We must give, too, a proper consideration to the practical importance of the ferro-alloy industry which has already begun operations on the Pacific Coast. Furnaces in Shasta county which were producing pig iron are operating on ferromanganese, ferrosilicon and ferrochromium, and they are at present operating successfully. There is a possibility, it seems to me, of firmly establishing those industries on this coast.

J. W. BECKMAN.—I take exception to what Professor Richards has said about ferromanganese. It is not possible to establish the ferromanganese industry here with an electric furnace. It might be with a blast furnace but not with an electric furnace.

D. A. LYON, Salt Lake City, Utah.—I would like to call attention to the fact that in this paper we do not state that it will never be feasible to establish electrochemical and electrometallurgical industries on this coast, and that the establishment of these industries should not be considered, but have stated our views as regards the possibility of electrometallurgical industries becoming large consumers of electric power here on the Pacific Coast at the present time.

As to the tonnage of steel used on this coast; the day before yesterday I was talking to a prominent iron and steel metallurgist of this country in regard to the possibility of producing steel here in the West, and he remarked that it would be practically impossible for any one plant to turn out all the different shapes that would be required to meet the demand, and that while the total tonnage of steel used on the Pacific Coast is comparatively large, there is not a large tonnage used of any particular shape or size. Such being the case, there is no demand at the present time for a plant producing a large tonnage of iron and steel. I think the gentleman was right and must therefore disagree with those who believe that there would be a market for the output from a large plant at the present time.

The Electric Furnace in the Foundry

Discussion of the paper of WILLIAM G. KRANZ, presented at the San Francisco meeting, September, 1915, and printed in *Bulletin* No. 101, May, 1915, pp. 927 to 930.

M. PETINOT, Niagara Falls, N. Y. (communication to the Secretary*). —I have read this paper with considerable interest and agree perfectly with Mr. Kranz that steel for castings made in an electric furnace is not necessarily, because of that fact, all good steel, and also that the qualities which he mentions, such as absence of segregation, elimination of sulphur, great tenacity, etc., can be obtained in an electric furnace if operated under proper metallurgical conditions.

An unusually low content of sulphur in steel for castings is not, of course, an absolute necessity and the range of this element usually obtained in steel made in basic open-hearth furnaces is sufficiently low to meet all physical requirements.

The main factor in making steel for castings, regardless of the process, is the complete deoxidation of the bath to prevent the formation of

* Received Sept. 13, 1915.

segregation and also increase the physical qualities of the steel. Such complete deoxidation is accomplished in basic electric furnaces by holding the metal sufficiently long in the presence of a blanket of carbide of calcium, as in the Heroult or Stassano process, or by the addition of 50 per cent. ferrosilicon, as in the Gerod or induction furnace process. However, if steel is so completely deoxidized in the furnace that when, as pointed out by Mr. Kranz, an addition of aluminum or titanium is made the added metal will not oxidize but will be found in the metallic state in the steel, there can be no doubt that such steel will reoxidize during the tapping of the furnace and also during the teeming from the ladle into the molds. While Mr. Kranz has undoubtedly through inadvertence failed to mention the fact, it is probably the practice at his plant to add aluminum in the ladle during the tapping of the furnace to completely deoxidize the steel.

Even admitting that an electric furnace properly controlled must give a more completely deoxidized product in the furnace than it is possible to obtain in such oxidizing processes as open hearth and converter, I still question its great advantage if it is still necessary to complete the deoxidation in the ladle.

In the accompanying schedule I have reproduced Table No. 1 showing the comparative properties of electric and basic open hearth steel as printed in Mr. Kranz's paper, and also the results of samples from six heats of basic open hearth and converter steels of approximately the same chemical composition.

These six analyses are representative of the quality of thoroughly deoxidized steel produced under good foundry practice and it will be noted that the physical properties compare very satisfactorily with those of the electric steel reported by Mr. Kranz.

Mr. Kranz mentions that "absence of segregation and oxides has been firmly established by a great many investigations in our chemical and physical laboratories" and gives as a proof of this fact that when ferro-titanium or aluminum is added to their electric steel either of these metals will be found as such in the steel, indicating that no oxides were present which would cause their oxidation.

I am inclined to believe that many metallurgists will be interested in obtaining full details in reference to this particular point made by Mr. Kranz. Personally, I have made hundreds of heats of steel in the same type of furnace which he is using. In many of these heats 0.10 per cent. titanium was used and in no case have overtraces of titanium been found in the finished casting.

At other times I have used aluminum in the ladle and occasionally added it into the stream of steel as it flowed from the ladle into the molds, particularly at the end of the operation because this addition of aluminum was judged necessary to insure the soundness of the castings.

In no case has aluminum been found in the finished casting, but in every case where an examination has been made for the presence of alumina, the oxide of aluminum has been established both by chemical and microscopical means.

I am of the opinion that what Mr. Kranz has reported as aluminum in the steels, which he has treated with that metal, is in reality alumina, and I believe that more complete information on this particular point will interest metallurgists generally.

TABLE I

	Elastic Limit	Ultimate Strength	Per Cent. Elong.	Per Cent. Red.	C	Mn	Si	S	P	
Electric steel.....	36,400	65,300	36.0	55.0	0.24	0.52	0.25	0.019	0.010	} Referred to by Mr. Kranz.
O. H. basic.....	34,800	63,000	29.5	35.0	0.23	0.53	0.24	0.038	0.011	
O. H. basic.....	40,900	70,400	35.9	56.3	0.24	0.55	0.30	0.020	0.020	
O. H. basic.....	36,200	64,200	35.9	61.3	0.23	0.51	0.35	0.020	0.012	
O. H. basic.....	35,800	68,300	35.9	61.4	0.23	0.77	0.34	0.027	0.012	} Test piece unannealed.
O. H. basic.....	38,700	61,600	35.0	54.0	0.19	0.56	0.22	0.027	0.014	
Converter steel....	38,000	69,200	32.0	53.6	0.21	0.76	0.37	0.055	0.041	
Converter steel....	40,500	71,000	32.0	52.8	0.21	0.68	0.34	0.052	0.044	

CARL H. BOOTH, Chicago, Ill.—Some people dwell a great deal on the superior quality of electric metal. I have believed that it is practical to make as good a quality of steel in other types of steel-melting equipment, providing skilled handling is obtained, but present-day experience indicates that the electric furnace is coming into general use because of the cost factor. The fact that you can get increased temperatures in an electric furnace makes it practical to make very thin castings, very nice delicate ones; it is ideal for small castings. Those are features which make it a highly attractive form of melting equipment, but the principal point and the one that will decide finally for those who are making steel and want to adopt some other form of melting, will be the cost of the manufacture of the steel itself. With other types of melting equipment you can get steel equally good. It will cost more to do so. I think that one point ought to be brought out here. It is well known that in Europe most of the high-grade steels are made in electric furnaces. That has probably led to the assumption that electric product is always better. As an argument for the salesman it is very nice to be able to talk that way. We really believe that it is easier to make a good metal with the electric furnace, but we do not say that it is not equally possible to make as good with other types of furnaces. The commercial question is the cost. With electric furnaces of proper design the cost can be made lower than the cost of melting steel with fuel furnaces, and it is this lower cost of steel production that is forcing the installation of electric steel furnaces.

J. W. RICHARDS, South Bethlehem, Pa.—We should not overlook the fact that there is a type of material that can be made by the electric furnace which cannot be made by the other. I refer to a quality of steel which is intermediate between open hearth steel and crucible steel. In practice there is a big gap in the quality of those two materials, and also in the price. It is possible to produce with an electric furnace, at an intermediate price, an intermediate quality of steel which fills a new field.

I have visited many steel casting plants in Europe, particularly in Norway and Sweden, and I am convinced that Mr. Kranz's statement properly represents the possibilities of the electric furnace. I think Mr. Booth is too modest in saying that you can obtain the best qualities of steel by other ways. The electric furnace competes with the very best quality of steel which is being made in the crucible. There are numerous plants in Germany that have torn out their crucible equipment and have substituted the electric furnace. Old established crucible steel plants there are making their finest qualities of steel, alloy steel and carbon steel, with the electric furnace. It is all a matter of price.

H. W. LASH, Cleveland, O.—I have always understood that the electric furnace offered advantages in the way of purity. In the open-hearth furnace there is more or less sulphur to deal with in the fuel, and of course oxygen is always present.

I have talked with Mr. Petinot and he showed me many results of electric made steel where they had almost eliminated the phosphorus acid sulphur and produced a steel that was very low in manganese and still worked well in the forge, largely because of the absence of oxygen.

CARL H. BOOTH.—My thought is that if you are willing to spend money enough you can take almost any furnace, whether electric or not, and the man who knows how could make steel, and make it good. Each different type of furnace has its advantages and disadvantages. The open hearth especially has disadvantages because of the conditions in the furnace. The electric furnace has a big advantage because of its ideal conditions, conditions which are under your control at all times. The atmospheric conditions are so good that you can do almost anything you want to do. Our thought has been to make the electric furnace so simple and so efficient that the cost of making these different steels will be cheaper than those made in any other way. The real advantage after all that will contribute in making the use of the electric furnace more general than the use of other types of equipment will be not only that it can make a good steel—it will have to compete on that basis just the same—but the question will be as to whether or not it can make steel cheaper than any other way. The earlier types of electric furnaces were so expensive to operate that they were largely used in the manufacture of

quality steel, principally high-grade alloy steels, for which the electric furnace is ideally suited, and for which a high price could be obtained. In Europe today the electric furnace is being used extensively for this type of service in place of crucible melting. Only in recent years, however, have electric furnaces been adopted for the manufacture of the cheaper grades of steel. There are electric furnaces running today in the United States which are producing steel at a lower cost than can be obtained with any other type of melting equipment for the same daily tonnage.

J. W. RICHARDS.—There is one point which needs to be cleared up. The crucible process can make the highest quality of steel, equal, let us say, to the best made in the electric furnace, but it requires the highest quality of raw material. It is possible for the electric furnace to make steel of that same quality from impure raw materials which cannot be used in the crucible at all. You cannot refine in the crucible. You can refine and make the highest quality of steel in the electric furnace, from what crucible steel makers would call low-grade material. That fact alone has sounded the passing of the crucible process.

The Commercial Production of Sound, Homogeneous Steel Ingots and Blooms

Discussion of the paper of EMIL GATHMANN, presented at the San Francisco meeting September, 1915, and printed in *Bulletin* No. 104, August, 1915, pp. 1485 to 1492

H. W. LASH, Cleveland, O.—I have found in practical experience that it is not objectionable to have some blow-holes in ingots. They enlarge during solidification and have a tendency to cause the steel to rise. If they are of fairly small size they weld up quite readily, especially where they are some distance from the surface. In that way a pipe, which is much more detrimental than a few small blow-holes, is prevented. There is a very great difference in the formation of blow-holes in steel. You get one condition in soft steel and entirely another condition in rail carbon, or the still higher grades. The figures that are given in Mr. Gathmann's paper I imagine are taken from the higher carbons, as it is very seldom that piping occurs in soft steel.

J. W. RICHARDS, South Bethlehem, Pa.—This contribution is very welcome, and I understand that the process of Mr. Gathmann is in practical use and has proven successful. I believe the experience of steel makers is that an ingot which has a pipe is of more solid structure, and therefore takes a great deal more work in the rolls and in mechanical treatment, than an ingot which has some blow-holes. The density of the piped steel is undoubtedly increased. My opinion is that it is un-

doubtedly best to try to completely deoxidize or kill the steel, and then try to overcome the secondary difficulties which are brought in by having a denser and more compact metal. If you completely deoxidize you know where you are at. If you leave some blow-holes in you may at times leave more than you desire, at others less. The blow-holes may come a little nearer to the surface of the ingot than you wish, and in reheating cause surface imperfections. I therefore think Mr. Gathmann's thesis is sound. But, in carrying it out in practice, you have to use some such device as Mr. Gathmann has invented to prevent a deep pipe which perhaps may go down to the center of the ingot. What he has succeeded in doing marks, I think, a distinct advance in the production of sound steel.

Conversion Scale for Centigrade and Fahrenheit Temperatures

Discussion of the paper of HUGH P. TIEMANN, presented at the San Francisco meeting September, 1915, and printed in *Bulletin* No. 102, June, 1915, pp. 1247 to 1248.

J. W. RICHARDS, South Bethlehem, Pa.—I have found in practice that among the many ways of figuring from one scale to the other, the following is the easiest to do mentally: Double the Centigrade figure, deduct one-tenth from this (you then have the Fahrenheit degrees above 32°), and finally add 32. Reversely, start by subtracting 32 from the Fahrenheit degrees, then take half, and to this add one-ninth its value.

A curious transformation was discovered by D. H. Childs.¹ To the Centigrade degrees add 40, take five-ninths, subtract 40. To the Fahrenheit degrees add 40, take nine-fifths, subtract 40. The rule is identical for the two cases, except for the inversion of the factor, and it is an interesting bit of arithmetic to show that the transformation is mathematically correct.

HUGH P. TIEMANN, Pittsburgh, Pa. (communication to the Secretary*).—This scale was gotten up primarily for personal use and for this purpose was kept on my desk where it would be readily available for one conversion or for a number. For isolated conversions required at considerable intervals of time, however, the ordinary formulæ worked out by pencil and paper should be practically as quick.

If my recollection is correct, there was some correspondence in the *Iron Age* recently in regard to so-called "quick methods" for conversion of temperatures, but it seemed to be the general conclusion that the standard formula: $F = \frac{9}{5}C + 32$, etc., is about as satisfactory as any, and

¹ *Metallurgical and Chemical Engineering*, vol. xi, p. 230 (May, 1913).

* Received Oct. 13, 1915.

that other methods are nearly, if not quite, as long, and if done mentally would offer the same chance for error.

Concluding his remarks, Professor Richards refers to "a curious transformation." While I do not pretend to be a mathematician, it seems to me that the explanation for the method in question is very simple:

It depends upon the fact that the respective zeros of the two scales are not at the same temperature, and that the same reading is secured only at -40° . The method in question, therefore, consists in (a) correcting a given reading so that it will be based on this common point (adding 40); (b) converting from one kind of degree into the other (multiplying by $\frac{5}{9}$ or $\frac{9}{5}$, respectively); and (c) restoring the new reading to the basis of its respective zero (subtracting 40).

The Thermal Insulation of High Temperature Equipment

Discussion of the paper of P. A. BOECK, presented at the San Francisco meeting, September, 1915, and printed in *Bulletin* No. 104, August, 1915, pp. 1539 to 1550.

LAWRENCE ADDICKS, Douglas, Ariz.—It does not seem to me that the rôle played by carrying heat away from one surface to another has been brought out. A cellular construction prevents the motion of air currents. Also in the installation on the top of a reverberatory furnace, that Mr. Krieger mentions, where it stopped the cooling, there is besides the question of overheating. Another complex question, entirely apart from the necessity of not carrying a brick above a certain temperature, is that of not allowing acids and bases to come together at too high a temperature.

J. W. BECKMAN.—What cements together the individual cells in the insulating material? Also, how does the material behave in the presence of acids and alkalies, both strong and weak?

A. H. KRIEGER, New York, N. Y.—The material is formed or composed of myriads of lace-like skeleton remains of diatoms visible only under a powerful microscope. The diatom lives in salt and fresh water. On the Pacific Coast there must have been an abundant life of the diatom, because the beds, still in existence in Southern California, are fully 1,000 ft. through, and are hard and strong, standing a pressure of more than 400 lb. per square inch. The beds must have been many ages in forming and the diatoms have become firmly packed together. The samples shown are typical of the entire deposit as it was formed at the bottom of the sea. Once this structure has been destroyed, it is impossible to reconstruct it without artificial bonding materials. With the exception

of hydrofluoric acid, acids do not affect this material. Strong alkalies dissolve it.

J. W. RICHARDS, South Bethlehem, Pa.—This paper has interested me very much, and must interest all those who use furnaces and are concerned with saving heat. A general statement in connection with fuel-fed furnaces is that from 50 to 90 per cent. of the heat generated passes away with the gases from the furnace, while from 10 to 50 per cent. of the heat generated is lost in conduction or radiation from the walls. There is, then, the possibility of saving from 10 to 50 per cent. of the heat which is generated in the furnace by proper heat insulation of the walls. In electric furnaces, however, the problem is still more important. The gases which are generated in the electric furnace will carry out of the furnace only from 10 to 50 per cent. of the energy generated within the furnace, and the loss from radiation may amount to from 50 to 90 per cent. of the total energy put into the furnace. Therefore, the study and proper utilization of these heat-insulating materials concerns from 50 to 90 per cent. of the energy which is used for running electric furnaces, and is one of the most important items upon which the furnace manager can experiment and study.

HENRY KREISINGER AND J. F. BARKLEY, Pittsburgh, Pa. (communication to the Secretary*).—At the experiment station of the Bureau of Mines in Pittsburgh, Pa., is a special furnace in the construction of which were used materials prepared from the Sil-O-Cel or infusorial earth, as it is commonly called. The furnace is a part of an apparatus designed for the study of heat transmission in steam boilers. The furnace is equipped for burning natural gas, and on some of the tests a temperature of 2,400°F. is maintained. Fig. 1 shows the construction of the furnace walls and the materials used in the construction.

The material marked firebrick is a high-grade firebrick known by the trade name "Benezet" and made by the Harbison-Walker Refractories Co.

The material marked "cork brick" and the "Nonpareil block" are the product of the Armstrong Cork Co. The cork bricks are made of powdered infusorial earth, clay, and ground cork. When the brick is burned the cork burns out and leaves a red porous brick very much like ordinary red brick in appearance but much lighter. The main ingredients of nonpareil blocks are powdered infusorial earth and asbestos fiber. It is not burned and looks very much like the ordinary magnesia pipe covering. The part of the bottom marked common brick is made of common red brick.

As shown in Fig. 1, the walls are solid, without any air spaces, the

* Received Sept. 16, 1915.

after having been found undesirable for high-temperature insulation. In order to obtain data for the computation of the quantity of heat dissipated from the furnace, also to obtain the figures of the comparative heat-insulating qualities of the different materials, the temperature drop through the walls was determined. For this purpose many thermocouples were imbedded to various depths and at various places into the walls. The locations of the junctions of these couples are shown by the small

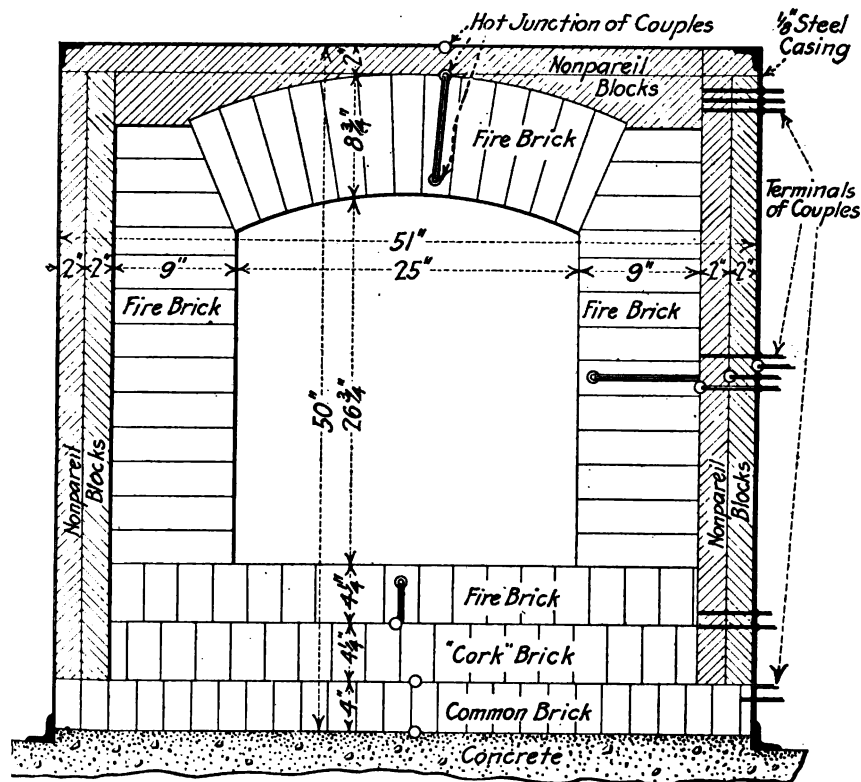


FIG. 1.—CONSTRUCTION OF FURNACE WALLS AND MATERIALS USED.

circles in Fig. 1. The couples subjected to temperatures up to about 2,000°F. were of copper and constantan. For temperatures above this point, Hoskin thermocouples made of nickel and nickel-chrome were used. All readings were taken with a Leeds Northrup portable potentiometer.

Fig. 2 shows the temperature fall through the bottom, through the top, and through the side of the furnace. The drop through the bottom is shown for four different furnace temperatures. The drop through the top is shown for two furnace temperatures, while the drop through the side

for one furnace temperature only. As shown in Fig. 2 the temperature drop through the cork brick and through the nonpareil blocks is several times as great as the temperature drop through the firebrick or common red brick. This great difference in the temperature drop indicates that the cork brick and the nonpareil blocks are much better heat insulators than firebrick or red brick. By using about 4 in. of cork brick or nonpareil

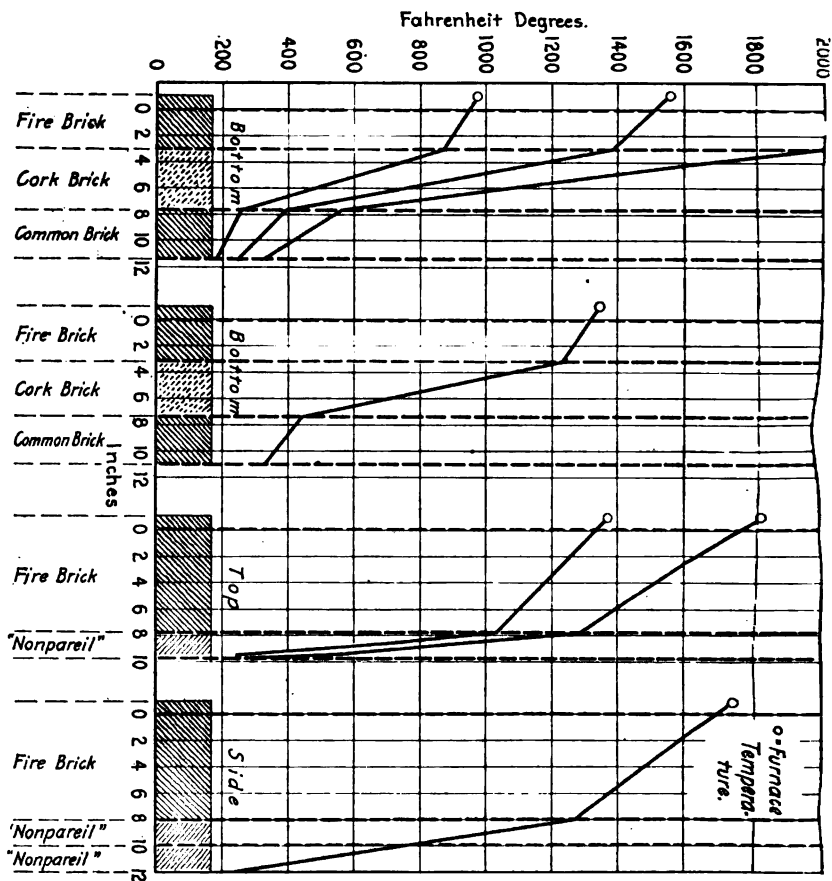


FIG. 2.—TEMPERATURE DROP THROUGH FURNACE WALLS.

blocks in the construction, a 12-in. wall is made as good a heat insulator as a 30-in. wall made entirely of brick or containing air spaces. The advantage of the 12-in. wall constructed as shown in Fig. 1 is that it is cheaper and occupies less space than a 30-in. wall built of solid firebrick. Another advantage of the 12-in. wall, especially for experimental purposes, is that its heat capacity is much smaller, and therefore equilibrium conditions are reached in a much shorter time than would be the case with the

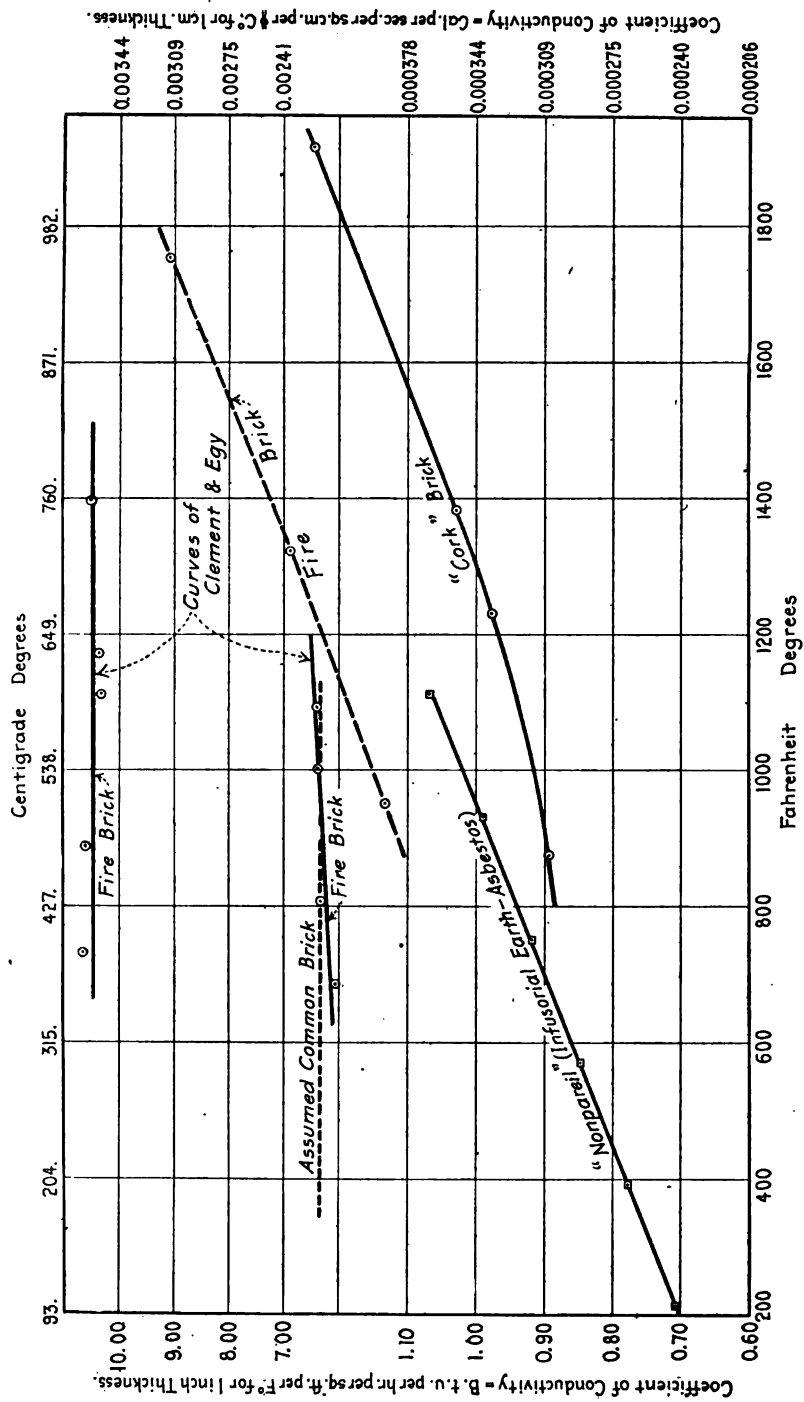


FIG. 3.—HEAT CONDUCTIVITIES OF DIFFERENT MATERIALS USED IN FURNACE CONSTRUCTION.

30-in. firebrick wall. This is of great importance in the investigations for which this furnace is used as it saves much time in heating up the apparatus. The disadvantages of the 12-in. wall for some types of apparatus might be its smaller strength.

The heat conductivities at various temperatures of the materials used in the construction of the furnace are given in the curves of Fig. 3. The abscissas are temperatures at which the conductivities were obtained; the ordinates are the conductivities. Attention is called to the fact that there are two ordinate scales on the chart; the lower scale applies to the nonpareil block and the cork brick; the upper high scale applies to the firebrick and common brick. The curve for the nonpareil blocks is plotted on absolute values obtained at the research laboratory of the Westinghouse Electric & Manufacturing Co. at Pittsburgh, Pa. The conductivity of the cork brick was computed from the data of Fig. 2 with conductivity of common brick assumed at 6.4 as a basis. This value of 6.4 was found to be the average of the results of various investigators. The curve of the cork brick is only relative to the assumed value of common brick; the curve may be either too high or too low, depending on whether the assumed conductivity of the common brick was too high or too low.

The curve of the conductivity of the firebrick was obtained from the data of Fig. 2 and is also only relative to the assumed conductivity of common red brick. The conductivity of firebrick computed from Fig. 2 varies greatly and the curve shown in Fig. 3 is only rough approximation. There is perhaps no exact value that will apply to one particular grade of firebrick; each individual brick has its own conductivity, differing perhaps many per cent. from other bricks of the same grade. To show the great variation in the conductivity of different firebrick the two curves of Clement and Egly are added to Fig. 3. The two curves were taken from a bulletin of the Engineering Experiment Station of the University of Illinois. According to the author's description, the firebrick test piece with which the lower curve was obtained was of reddish-brown color and of medium coarse structure. It contained very small pieces of white gravel throughout the mass. The higher curve was obtained with a test piece a little coarser in structure and of brown color. It contained a very small amount of gravel. As indicated by the two curves, the conductivity of one piece was about twice as high as that of the other.

The highest temperature maintained in the furnace is far below the melting of the firebrick lining so that it was not necessary to take any precautions on that account. In furnaces where much higher temperatures are maintained, good insulation may prove destructive to the fireproof lining. This is particularly true if the furnace gases carry slag in suspension. The slag settles on the furnace lining and causes the firebrick

o melt at a somewhat lower temperature than it would if no slag were present.

The brick used for lining furnaces are usually mixtures of several compounds and therefore have no definite melting points. Some of the compounds melt at lower temperatures than others, so that the change from the first signs of softening to complete melting-usually covers a wide temperature range. The following are the melting points of several kinds of brick used in furnace linings. The values are taken from Bulletin No. 10 of the Bureau of Standards, Melting Points of Firebrick, by C. W. Kanolt. The author defines the melting points as the lowest temperature at which a small piece of the firebrick could be distinctly seen to flow.

TABLE I.—*Melting Points of Various Bricks*

Name of Brick	Melting Point °F.		
	Low	High	Average
Fire clay.....	2,831	3,137	3,002
Bauxite.....	2,849	3,245	3,080
Silica.....	3,092	3,101	3,090
Chromite.....			3,722
Magnesia.....			3,929

Manufacture and Tests of Silica Brick for the Byproduct Coke Oven

Discussion of the paper of KENNETH SEAVER presented at the San Francisco meeting, September, 1915, and printed in *Bulletin* No. 105, September, 1915, pp. 1913 to 1927.

J. W. RICHARDS, South Bethlehem, Pa.—This paper concerns a very large industry, and an important detail in that industry. Within sight of where I live in Bethlehem, Pa., are coke works in which \$2,500,000 was lost by the use of improper bricks in the coke ovens. This was the fundamental reason of the failure of the plant. The old ovens have all been torn out and replaced by new ovens built of proper bricks. This may emphasize the commercial importance of this subject.

When speaking about coke-oven brick, the first thing that comes to my mind is how important it would be if it were possible to get a brick of better heat conductivity. The heat for coking coal has to go through the thickness of the brick. The silica brick used has low heat conductivity as compared with some other refractories; magnesite brick probably has four times its heat conductivity. The coke oven needs bricks of high heat conductivity. Any process which would increase the heat conductivity of silica brick would undoubtedly shorten the coking process.

WILLIAM A. WILLIAMS, San Francisco, Cal. On p. 1926, the tests of bricks that have been burned once, twice and three times, and the width of the brick in inches, is given. From the first burn to the second there seems to be an expansion, and from the second to the third a remarkable contraction.

KENNETH SEAVER.—Part of those tests were made on what we term 9-in. straights and part were split bricks, and not on the same brick. It is confusing the way it is expressed in the paper.

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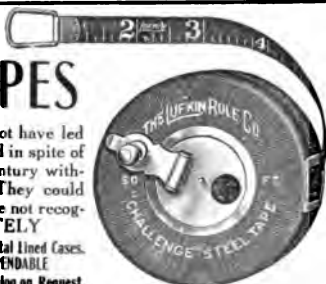
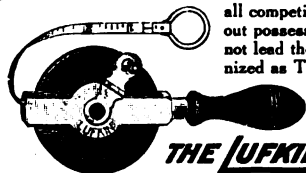
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These consist of a paper shell in one end of which is fastened the powder or igniting composition and the insulated copper or iron wires, the other end being open. The paper shells are made in the shape of a truncated cone, in order that a blasting cap may be inserted into the shell and be more tightly wedged in the farther it is inserted. It is, therefore, not necessary to crimp the blasting cap in the shell.

Without the blasting cap they are used to ignite blasting powder electrically, but can be used to detonate dynamite by using them with a blasting cap. By their use, it is possible to ignite a charge of blasting powder in the center, the ideal position, which secures the maximum efficiency of the explosion, impossible with either fuse or miner's squibs.

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They are made with either insulated copper or iron wires. The lengths of the copper wires range from 4 to 12 ft. The lengths of the iron wires range from 4 to 8 ft.—*Du Pont Magazine*.

Trade Catalogues

Ingersoll-Rand Co., 11 Broadway, New York, N. Y., has issued two new booklets, Forms 76 and 9201. The former is an 80-page catalogue on the subject of air lifts in which the air-lift system of pumping is explained in detail. To those interested in the pumping of large quantities of clean water, this booklet will be found of value, since the essential facts are well presented.

The second book contains 128 (6 by 9-in.) pages describing Calyx core drills. The principal function of the core drill is the determination of the character, order, thickness and extent of the materials beneath the earth's surface, by means of cylindrical cores which it extracts. In coal and metal mines, stone quarries, contract work, or for canal or tunnel developments projected, the core drill is used and takes what is otherwise an unknown quantity and converts it into an absolute certainty. By determining the actual conditions in advance, it is a medium to save thousands of dollars in machine equipment, underground workings, etc., that might otherwise be expended only to find a barren property. The booklet is illustrated, and gives the size and capacities of the different types showing the apparatus in operation in various fields.

Thermit Process.—Goldschmidt Thermit Co., 92 West St., N. Y. Several well-illustrated descriptive catalogues and booklets on the thermit process for repairing steel and iron rolls, pinions, locomotive repairs; in other words, the application of the Thermit Welding process in all kinds of work.

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"Duriron."—The Duriron Castings Co., 92 West St., N. Y. A 16-page illustrated catalogue descriptive of its acid-, alkali- and rust-proof metal. Data on chemical tests, physical characteristics and general information are given.

Blasting Powder and Accessories.—E. I. DuPont de Nemours Powder Co., Wilmington, Del. Two catalogues, illustrated, 63 and 126 pp. respectively, descriptive of blasting powder, its manufacture, and instructions for its use. Many "don'ts" should prove of value to even the experienced "blaster." The second catalogue covers blasting machines, electric fuse, and squibs, leading wire, thawing kettles, caps, and other equipment for use in blasting.

Instruments of Precision.—W. & L. E. Gurley, Troy, N. Y. Complete catalogues and bulletins descriptive of civil engineering instruments, such as Light Mountain Transits, Explorers' Levels, and Transits, Y Levels, Beaman Stadia Arcs, Pocket Compass, Miner's or Dip Compass, Cox Stadia Computer, Current Meters, Automatic Water Stage Registers, Leveling Rods, Sketching Cases for Civil and Military Engineers and Surveyors. Also a booklet on "How to Use and Adjust the Architects' Level, Drainage Level and Builders' Transit." All well written and illustrated.

"Wire Rope" and *"Aerial Wire Rope Tramways,"* are the titles of two booklets, 26 and 94 pp. respectively, published by A. Leschen & Son, Rope Co., St. Louis, Mo., which illustrate and describe wire rope construction, grades of wire rope, and give information on its use and care, and other data of interest to users of wire rope for ordinary purposes and tramways.

Conveyor Belts, Chains, Sprocket, and Elevators, Ore, Coal and Coke Crushers, Feeders and Elevators, and Repair Parts.—The Robins Conveying Belt Co., 15 Park Row, New York. Bulletins Nos. 47, 48, 49, 51 and 52 illustrate and describe the products and method of this company. They contain interesting data for those having conveying problems.

Mining Equipment.—Chalmers & Williams, Inc., Chicago Heights, Ill. Several sections of Catalogue and descriptive literature on complete equipment for concentrating, cyaniding, rock crushing, and cement plants, amalgamating, ore reduction, rock crushing, tube and ball mills, etc. All well written and profusely illustrated.

Metallurgical Clay Goods.—The Denver Fire Clay Co., Denver, Colo. Bulletin No. 100 (18 pp.) illustrates and describes muffles, crucibles and scorifiers manufactured by this company. Valuable information is given on the care and uses of crucibles. Another bulletin describes and illustrates the Case low-pressure oil forge for heating drill steel.

Switchboard Panels for Railway Service.—General Electric Co., Schenectady, N. Y. Bulletin No. 47010 for August describes and illustrates 600- to 1,500-volt direct-current switchboard panels, 90 or 99 in. high. Data on general application, operation and construction, as well as capacities, etc., are given.

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Monel Metal.—Supplee-Biddle Hardware Co., Philadelphia, Pa. Bulletin No. 7 describes an unusually large valve for superheated steam service, in which are used cast Monel metal seat rings. The bonnet has a "pack-when-open" bushing of the same metal. Other data on Monel metal are given.

Reels for Band Chains and Metal Tapes.—Keuffel & Esser Co., Hoboken, N. J. Circular describes two new reels for band chains and flat-wire tapes. Illustrated.

Lettering Templets.—Keuffel & Esser Co., Hoboken, N. J. Circular. Illustrated. Describes templet for lettering.

Engineers Transits.—W. and L. E. Gurley, Troy, N. Y. New 8-page booklet describing the new and improved Gurley engineers' transits—with one-piece trussed standard. Well written and illustrated.

Safety Fuse and Fuse Lighters for Deep-Well Blasting.—The Ensign-Bickford Co., Simsbury, Conn., has issued several catalogues and booklets, illustrating and describing, with practical instances, its safety fuse and lead-spitter fuse lighters.

Fans.—Jeffrey Mfg. Co., Columbus, Ohio. Booklet entitled Jeffrey Fans for Mine Ventilation. Illustrates and describes its line of centrifugal fans.

Trade Note

The Sullivan Machinery Co. announces the removal of its St. Louis branch sales office from 705 Olive St. to suite No. 2006 Railway Exchange Building, where larger and more desirable quarters have been prepared. This change will enable the company to give still better attention to the requirements of its customers in the territory served by the St. Louis office.

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ALPHABETICAL LIST OF ADVERTISERS

	Page
Albany Lubricating Co.	5
Allis-Chalmers Manufacturing Co.	12
Beer, Sondheimer & Co.	7
A. S. Cameron Steam Pump Works	11
Crocker-Wheeler Co.	12
Denver Fire Clay Co.	12
Denver Rock Drill Co.	12
Dorr Cyanide Machinery Co.	7
Du Pont Powder Co.	3
Ensign Bickford Co.. . . .	7
General Electric Co.. . . .	Outside Back Cover
Goodman Manufacturing Co.	13
B. F. Goodrich Co.	Inside Front Cover
W. & L. E. Gurley	1
Harbison-Walker Refractories Co.	14
Heller & Brightly	1
Illinois Zinc Co.	14
Ingersoll-Rand Co.	3
Jeffrey Manufacturing Co.	Inside Back Cover
Lead Lined Iron Pipe Co.	15
A. Leschen & Sons Rope Co.	15
Lufkin Rule Co.	5
Mashek Engineering Co.	15
Robins Conveying Belt Co.	15
John A. Roebling's Sons Co.	Inside Back Cover
P. H. & F. M. Roots Co.	16
Chas. A. Schieren.	5
Sullivan Machinery Co.	9
Terry & Tench Co.	16
Traylor Engineering & Manufacturing Co.	9
Union Iron Works	1
L. Vogelstein & Co.	19
Vulcan Iron Works.	11
Westinghouse Electric & Mfg. Co.. . . .	17
Weston Electrical Instrument Co.	5

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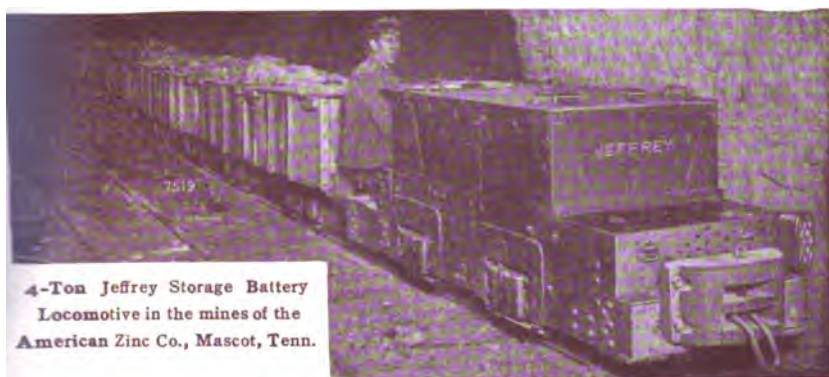
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ALPHABETICAL LIST OF ADVERTISERS

	PAGE
Albany Lubricating Co.	5
Allis-Chalmers Manufacturing Co.	12
Beer, Sondheimer & Co.	7
A. S. Cameron Steam Pump Works	11
Crocker-Wheeler Co.	12
Denver Fire Clay Co.	12
Denver Rock Drill Co.	12
Dorr Cyanide Machinery Co.	7
Du Pont Powder Co.	3
Ensign Bickford Co.	7
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Goodman Manufacturing Co.	13
B. F. Goodrich Co.	Inside Front Cover
W. & L. E. Gurley	1
Harbison-Walker Refractories Co.	14
Heller & Brightly	1
Illinois Zinc Co.	14
Ingersoll-Rand Co.	3
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Lead Lined Iron Pipe Co.	15
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Robins Conveying Belt Co.	15
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P. H. & F. M. Roots Co.	16
Chas. A. Schieren.	5
Sullivan Machinery Co.	9
Terry & Tench Co.	16
Traylor Engineering & Manufacturing Co.	9
Union Iron Works	1
L. Vogelstein & Co.	19
Vulcan Iron Works.	11
Westinghouse Electric & Mfg. Co.	17
Weston Electrical Instrument Co.	5